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NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

TRANSACTIONS.

VOL. XLII.

1892–93.

EDITED BY M. WALTON BROWN, SECRETARY.

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NORTH OF ENGLAND INSTITUTE

OF

MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE,

OCTOBER 8TH, 1892.

MR. J. B. SIMPSON, PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the last General Meeting and reported the proceedings of the Council at their meetings of August 6th, September 24th, and October 8th.

INTERNATIONAL CONGRESS OF ENGINEERS, CHICAGO, 1893.

The SECRETARY stated that he would be pleased to receive the names of members who desired to communicate papers to the International Engineering Congress, to be held at Chicago, in 1893, during the World's Columbian Exposition.

THE FEDERATED INSTITUTION OF MINING ENGINEERS.

The SECRETARY reported the proceedings of the meeting of the Council held on September 7th, 1892.

The following gentlemen were elected, having been previously nominated:—

MEMBERS—

Mr. JAMES ANGUS, Mining Engineer, Radcliffe, Acklington, Northumberland.

Mr. ROBERT JOHN BROWN, Colliery Manager, Barakar, East India Railway, Bengal, India.

Mr. JOHN CURRY, Colliery Engineer, Rainton Colliery, Fence Houses.

Mr. PETER HARLE, Assistant Manager, Page Bank Colliery, Co. Durham.

Mr. ROBERT THOMAS MOORE, Civil and Mining Engineer, 156, St. Vincent Street, Glasgow.

Mr. WM. MORRIS, Jun., Colliery Manager, Waldridge Colliery, Chester-le-Street.

Mr. JAMES LESLIE SHAW, Civil and Mining Engineer, Somerset House, Whitehaven.

ASSOCIATE MEMBERS—

Rev. G. M. CAPELL, Passenham Rectory, Stony Stratford.

Mr. GEORGE ELLIOT COXON, Explosive Agent, Low Fell, Gateshead-upon-Tyne.

Mr. George Humphreys-Davies, F.S.I., Surveyor, Valuer, and Rating Expert, 8, Laurence Pountney Hill, Cannon Street, London, E.C., and 16, Deansgate, Manchester.

Associate—

Mr. Edward Charlton Scott, Surveyor, Rainton Colliery, Fence Houses.

Students—

Mr. Robert O. Brown, Mining Student, Elswick Collieries, Newcastle-upon Tyne.

Mr. Fred. King, Mining Student, Cramlington Colliery, R.S.O., Northumberland.

Subscribing Collieries—

Butterknowle Coal Company, Limited, Darlington.
Cowpen Coal Company, Limited, Newcastle-upon-Tyne.
Elswick Coal Company, Limited, Newcastle-upon-Tyne.
Westport Coal Company, Dunedin, New Zealand.

The following gentlemen were nominated for election :—

Honorary Member—

Prof. Henry Stroud, D.Sc., Durham College of Science, Newcastle-upon-Tyne.

Members—

Mr. Alexander Mathieson, Colliery Manager, Hetton Colliery, Carrington, near Newcastle, New South Wales.

Mr. Clayton E. J. Mitchell, Civil Engineer, 27, Mosley Street, Newcastle-upon-Tyne.

Mr. James Newton, Colliery Manager, Whitehaven Colliery, Whitehaven.

Mr. George Oldham, Engineer, 25, Western Hill, Durham.

Mr. T. A. Tate, Colliery Manager, Bunker Hill, Fence Houses.

Associate Members—

Mr. George Blaxland Atkinson, Timber Merchant, Maritime Buildings, Quay, Newcastle-upon-Tyne.

Mr. J. P. Cornett, Managing Director, Ford Paper Works, Hylton, Sunderland.

Mr. Arthur Patrick Dodds, Iron Merchant, 13, Dean Street, Newcastle-upon-Tyne.

Mr. Claude Bowes Palmer, Assistant Colliery Manager, Usworth Colliery, Washington, R.S.O.

Associates—

Mr. Thomas Berry, Overman, Bagnall's Houses, Swalwell, R.S.O.

Mr. Robert Carr, Under Manager, Usworth Colliery, Washington, R.S.O.

Mr. William Severs, Under Manager, Beamish View, *via* Chester-le-Street.

Students—

Mr. William Henry Mascall, Mining Student, 27, South Church Lane, Bishop Auckland.

Mr. George Wall Mining Student, South Church, Bishop Auckland.

DISCUSSION ON MR. W. J. ORSMAN'S "NOTES ON THE PRODUCTS AND TEMPERATURE OF DETONATION OF SOME HIGH EXPLOSIVES."

Mr. SAVILLE SHAW said the first point he desired to draw attention to was with reference to the products of detonation of roburite. In view of the interest which had lately been taken in the products of its explosion it would have been of advantage if the paper had given more detail as to the methods of collecting, measuring, and especially of analysing the gases evolved. It would be remembered that roburite was a mixture of dinitro-chloro-benzene with nitrate of ammonia, and when this was detonated they got as the products—carbon dioxide, nitrogen, steam, and hydrochloric acid; this was according to the theoretical equation. It was stated in the paper that "the presence of the chlorine compounds in the ultimate gaseous products tends to still further reduce the effect of the escaping heated gases." He would have liked Mr. Orsman's explanation on this point. The mean of twelve experiments showing what the ultimate gaseous products were proved them to consist exclusively of carbon dioxide and nitrogen (carbon dioxide 32 per cent. + nitrogen 68 per cent. = 100) thus allowing nothing whatever for the chlorine compounds referred to. If these compounds were present how was it that they were entirely overlooked in the analyses? Another point which was of interest to chemists was in regard to the volume of the evolved gases. In the paper 10 grammes of roburite were said to give 4,780 cubic centimetres of gas, but it was quite impossible to obtain such a large amount of gas from that quantity of roburite; the largest calculated proportion being 3,496 cubic centimetres. With roburite of the composition given by Mr. Orsman, 10 grammes would contain 8·6 grammes of nitrate of ammonia and 1·4 grammes of dinitro-chloro-benzene. If the whole of the nitrogen in the nitrate of ammonia were converted into free gas it would occupy 2,408 cubic centimetres at 32 degs. Fahr., and this amount added to the nitrogen from the dinitro-chloro-benzene, viz., 154·8 cubic centimetres, gives a total possible volume of gaseous nitrogen of 2,562·8 cubic centimetres. Similarly, the amount of dinitro-chloro-benzene used contains ·5 gramme of carbon, which converted into carbon dioxide would furnish 933·3 cubic centimetres of that gas. This volume added to that of the nitrogen gives the total amount of the two gases as 3,496·1 cubic centimetres as against the 4,780 cubic centimetres found by Mr. Orsman. Even if the evolved

gases had been measured at 59 degs. Fahr. instead of being reduced to 32 degs. Fahr. their volume would only be raised to about 3,688 cubic centimetres, and if the gas from the detonator, containing 15 grains of fulminate, were added to that, the total calculated volume would still be below 4,000 cubic centimetres. This error of something like 25 per cent. was somewhat important, considering the nature of the substances dealt with. No doubt the writer had some simple explanation of the apparent discrepancy, but at present he (Mr. Shaw) was quite at a loss to account for it.

Mr. Wm. Jas. Orsman, in reply, wrote that full details were given in the paper as to the methods of detonating the explosives and collecting the resultant gases. Having obtained these gases, bottled up in a cylinder, it was very easy to determine their composition by the ordinary laboratory methods. The chlorine present in the dinitro-chloro-benzene combines with the hydrogen to form hydrochloric acid, which exists as gas when the shot is fired, but on the cooling of the gases separates out with the condensed water vapour and collects at the bottom of the cylinder. As the analyses of the gases were made at the ordinary temperature, and were meant to show what products actually escaped into the air of a mine, no notice was taken of the amount of hydrochloric acid and water vapour formed. Similarly, no deduction was made for the volume of gas formed from the burning of the detonator fittings, which consist of paper and marine glue, and weigh between 2 and 3 grammes. No claim is made to give an exact analysis of the products from the explosives themselves, but to get a general idea of the nature of the gases formed when such explosives are in actual use down a pit. The principal aim of the whole series of experiments was to establish the fact that such attempts to lower the temperature of detonation of nitro-glycerine mixtures by the addition of carbonaceous matter, as in carbonite, means the production of the poisonous gas, carbon monoxide. With roburite, on the other hand, which contains an excess of nitrate of ammonia, no carbon monoxide is formed, and any further addition of nitrate of ammonia would only tend to still further lower the temperature of detonation.

The discussion then closed.

DISCUSSION UPON MR. HENRY PALMER'S "NOTES ON AN ELECTRIC TRANSMISSION PLANT AT EAST HOWLE COLLIERY," AND MR. D. SELBY BIGGE'S PAPER ON "THE PRACTICAL TRANSMISSION OF POWER BY MEANS OF ELECTRICITY, AND ITS APPLICATION TO MINING OPERATIONS."

Mr. Sydney F. Walker (Cardiff) wrote that a great many of the remarks he had made in the discussion upon Mr. Snell's paper* were equally applicable to Mr. Palmer's paper. The writer's objections, for instance, to the use of old wire-rope as either lead or return cable applied with their full force here, and in addition he noticed that Mr. Palmer had experienced some difficulty with the electrical resistance offered by the old wire-rope. This was a very important point, which should receive careful consideration. If many of the wires of the rope were broken, as the writer thought would naturally be the case before the rope was abandoned for mechanical purposes, the measured electrical resistance would be very much higher than the dimensions and weight would point to, as the continuity of each wire, or of a very large number of them, being broken, conduction through the mass of the rope would take place very largely across the oxidized surfaces of these wires, and such surface contacts always have a very high resistance. In addition to this, if it had been necessary to joint two pieces of old wire-rope together, the joint itself would add considerably to the resistance offered. The writer believed that if all items of cost were carefully reckoned it would be found cheaper to use two insulated copper cables than to use old wire-ropes for the return. As the writer pointed out in the discussion on Mr. Snell's paper, he did not like making the whole mine part of the circuit, as would be done if a naked return was used for any voltage that can be utilized either for lighting or power. The writer was not surprised therefore to find how great was the loss between the generator and the motor. No doubt a large part was due to the old iron wire-rope. The writer noted also that the actual return in work done, water raised, of the indicated horse-power in the driving engine was the lowest on record. This appeared to be due very largely to the small pipes that were used, but it showed how necessary it was to have all the factors before judging of the relative efficiencies of any two methods of transmission. It by no means followed, however, that because there was a heavy loss both in the cables and in the water-pipes

* *Trans. Fed. Inst.*, vol. ii., page 150.

that Mr. Palmer did wrong in adopting electricity, or even that he was working extravagantly. This question could only be decided by placing the cost of the additional horse-power which had to be furnished against the interest of the additional outlay on cables and water-pipes that would have been necessary to avoid these losses. One point appeared to the writer to come out very strongly in Mr. Palmer's paper, viz., the great convenience which attended the use of the electric method of transmission, and the speed at which the apparatus could be erected. This was always worth money, and had done more and would do more to spread the use of electrical apparatus of all kinds than any other quality it possessed.

Mr. Sydney F. Walker (Cardiff) wrote respecting Mr. Bigge's paper that he had not much to add to his remarks upon Mr. Snell's and Mr. Palmer's papers. As already mentioned, he did not think that the plan of enclosing the commutator within a gas-tight box, or of placing the brushes inside the commutator, was likely to prove more than a temporary expedient. The covers in either case would not remain gas-tight after being in use for some time, while the inconvenience of the arrangement was likely to give trouble. The best remedy for the danger of igniting gas from the sparks at the commutator of any dynamo would be the reduction of the spark below firing point rather than the enclosure of the commutator or of the brushes. It was now well understood that it was not a spark *per se*—any spark—that would ignite an inflammable mixture, but a spark which contained a certain definite quantity of electrical energy. This quantity appeared to be at least 20 watts, as far as experiments and observation have enabled the writer to determine, or one-third of the energy at present employed in a 16 candle-power incandescent lamps: consequently where it is necessary to use an electromotor in a gaseous atmosphere the motor should be so constructed as never to give rise to sparks having this amount of energy, under any possible conditions. This is perfectly practicable, at a cost not greatly exceeding that involved in making the armature gas-tight. Professor Silvanus Thompson found that blue sparks did not ignite gas; and Mr. Snell found that unless the copper of the commutator or brushes was heated to a certain degree gas could not be ignited. The writer's own experiments, carried out some years ago, and repeatedly confirmed, show that an electromotive force of at least 2 volts must be present across the gap over which the spark leaps, and that when the voltage is at this low figure the current passing in the spark—it is a true electric current, just as that which furnishes the arc lamp is—should be not less than 10 ampères. When the current is used to generate heat in a wire, according to the

writer's experiments, the wire must be quite yellow before it will ignite gas, if present. The time factor, of course, must be considered in this matter, as in all other engineering problems. Where the sparking is continuous, and the surrounding conditions are such that the atmosphere in the immediate neighbourhood of the commutator rises in temperature, it may ignite in time if the atmosphere is inflammable. But such conditions can never be present where a motor is used. It may be assumed that even at the face of the coal only occasional outbursts of gas are to be feared, which will be quickly diluted below the dangerous point by the ventilating air-current, so that the dangerous mixture could never remain long enough in the neighbourhood of the commutator to be heated in the manner described. Further, the revolution of the armature itself must and does create convection air-currents which would produce the same result. An interesting point here arises: suppose that a gaseous mixture be inadvertently enclosed within the cover of the commutator, would not this tend to become heated to the dangerous point, merely from the fact of its enclosure, notwithstanding the convection currents set up by the revolving armature? With reference to the question of sparking across the break when a cable is sheared by a fall of roof, the writer certainly admires the ingenuity with which the inventors, the Messrs. Atkinson, have worked out their non-sparking cable. He was present at Prof. Silvanus Thompson's lecture at Cardiff, last year, when he exhibited the Atkinson cable; it behaved perfectly when broken, there was no spark at the break, and the lead fuze behaved as it was wanted to, but when he proceeded to handle the cable again there was a spark between the broken ends of the cable, or between one end of the main cable and the central wire. As far as he remembered, Mr. Atkinson's explanation of this was quite correct, viz., that an accidental connexion was made. But the question arises: might not this same accidental connexion, or something similar, take place in the mine? It must be remembered that the main cable remains connected to one terminal of the dynamo, while the inner safety wire remains connected, through the motor or lamps, to the other terminal of the dynamo, so that should the break of the main cable be a jagged one, as may easily be, a fresh connexion and a fresh break may occur, giving rise to a spark. There is also the possibility of the wire fuze not acting; fusible cut-outs are unfortunately not reliable. The passage of the current through them, and the vibrations of the building where they are situated, tend to make them brittle, and cause them to "go" when they are not wanted. In order to avoid this, it is necessary to have the fuze wire stronger than otherwise would be necessary, so that it may not

be a constant source of trouble ; and this results in the fuze taking a long time to melt if it happens to be nearly new when called upon to act. In the writer's opinion if this cable is used, an electro-magnetic automatic cut-out should be employed in place of the fusible cut-out, and some such device as that shown in Fig. 1 should be inserted in the circuit of the main cable, so as to break the connexion with the dynamo immediately the cable is sheared underground. The idea of the apparatus is as follows :—A bent lever A, one arm of which forms a counterbalance weight, and the other carries a contact piece C, is inserted in the circuit; the mercury cup B, into which the contact piece C dips, being also part of the circuit. Where the current to be controlled is large, there are two contact pieces dipping into two mercury cups. The coils of the electro-magnet D are also in the same circuit, that of the cable to be cut off from the dynamo. A second lever E, with one arm very much longer and heavier than the other, engages with the toe-piece of the contact lever C, and its lighter arm carries an iron armature F, which faces the pole of the electro-magnet. The current, when the circuit is complete, passes from the dynamo round the coils of the electro-magnet D, thence to the mercury cup B, from there to the lever C and to the pivot or second mercury cup as arranged, and so on to the cable. As long as the current is passing round the electro-magnet the light end F of the lever is held down by the attraction of the pole of the electro-magnet for its armature, and the contact piece is free to rest in the mercury cup. The counterbalance A is also then acting by its weight to keep the contact piece C firmly in the mercury cup B. On the cable being broken below, the current ceases round the electro-magnet coils D, the armature F is released, and the weight of the opposite arm of the armature lever E causes the end next the contact piece to cant sharply upwards, knocking the toe of the contact lever up, and throwing the counterbalance A to the other side of the vertical line passing through its pivot. The contact lever revolves on its pivot, the counterbalance falling backwards, thereby breaking the circuit. In Fig. 2 is shown the electro-magnetic automatic cut-out which the writer has designed for use with electric lights and power circuits in mines. The arrangement is similar to that of the circuit breaker already described, but the office of the electro-magnet D is

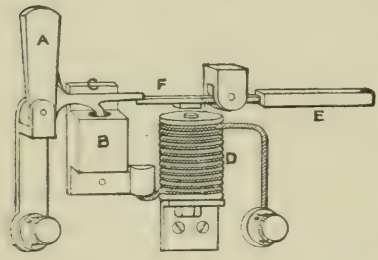


Fig. 1.

now to break the circuit on the arrival of a current whose strength is above a certain figure. The armature lever F in this apparatus has its two arms almost equal, the balance being on the side of the one E, which acts as a tripper. The armature is on that arm of the lever away from the contact lever, as shown, where it is faced by the pole of the electro-magnet. The current passes, as in the other apparatus, through the coils of the electro-magnet D to the mercury cup B, thence to the contact lever, and so on. When at rest, that is to say when the circuit is complete, the contact piece C rests in the mercury cup B, and the weight of the counterbalance F tends to keep it there. On the arrival of a current of the strength for which the apparatus is set, the armature F is pulled very sharply towards the pole of the electro-magnet D, carrying with it that arm of the lever. The opposite arm E of the lever gives a sharp cant to the contact lever C, which falls over, breaking circuit as before. The writer has also arranged the apparatus to work with an excess electro-motive force, and to complete a second circuit after breaking the ordinary working circuit if required. This apparatus acts with great promptness in such a case as the accidental contact of two wires connected to opposite sides of the dynamo; so quickly, indeed, that if, as often happens, a wire touches another and is drawn away, the spark between the two surfaces is very small indeed, though an ampère meter shows the passage of a very powerful current while the contact lasts. Electro-magnetic automatic cut-outs for use in case of short circuit, or over-running, if carefully employed, are of great service; but he does not recommend the use of the other apparatus except in special cases, as it tends to add to the complications at the switch board. He does not like fusible cut-outs, and only uses them in very special cases. The cable itself should be made strong enough to stand the shearing strains brought by heavy falls of roof, but the additional strength required for this purpose, over and above that of the cable as it would be turned out in the ordinary way, should be given by means of several longitudinal wrappings of some fibrous insulating substance, such as jute. The necessary strength can be thus obtained, and freedom from sparking thereby avoided at a very considerably less cost than would be incurred in the special form of cable designed by the Messrs. Atkinson, or in any external wire armour. Though steel and

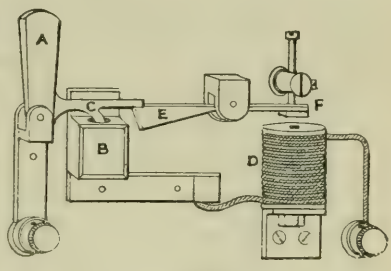


Fig. 2.

iron have very properly taken the place of hemp for ropes that are used both in mines and ships, it is mainly because they wear better and occupy a smaller space. But a cable for electric light or power has no appreciable wear, the additional space occupied by the jute wrappings are no inconvenience, and on the other hand the cable so made is handier in every respect. It is more flexible, more easily laid, and more easily jointed. Further, there is every chance of a cable so made stretching sufficiently, under a very heavy fall to avoid fracture. We are apt to forget that hemp has great mechanical strength if there is only plenty of it.

Mr. C. C. Leach (Seghill) said an electric pumping-plant had recently been put down at Seghill colliery. It is driven by a Robey vertical engine with a cylinder 9½ inches in diameter and 9 inches stroke, fitted with a Pickering governor. It is intended to work at 100 lbs. per square inch, although at present the governor reduces the steam down to 39 lbs. The fly-wheel, which acts as the belt pulley, is 3 feet 6 inches in diameter. A link leather belt, 6 inches wide and ½ inch thick, is used to drive with 13 feet centres. The Goolden dynamo is intended to produce 24 ampères, at 460 volts, when running at 780 revolutions per minute. It is fitted with a pulley 10 inches in diameter, and is said to produce 15 horse-power. The cables are, in all, 3,160 yards long of 7/16 copper, rubber-insulated; this length of cable is used for the distance from the dynamo at bank to the motor down the pit and back to the dynamo. The 11 horse-power motor is of the Goolden gas-tight pattern, equal to an output of 24 ampères at 400 volts when running at 1,050 revolutions per minute. The pump has three single-acting rams of 5½ inches diameter and 9 inches stroke, geared down by double helical teeth in the ratio of 24·87 to 1. The vertical head or lift from the water-edge to the delivery is at present 121·88 feet. The single suction-pipe is 31 yards long, 11·54 feet head, and 4 inches diameter, with faced joints. The rising main consists of two pipes, 4 inches in diameter, each 886 yards long, with 110·34 feet head, making a total length of 917 yards of pipes from the water-edge to the delivery. The following tests have been made of the plant :—

Observations.	Sept. 14, 1892. Average of Seven Tests.	Sept. 23, 1892. Average of Seven Tests.
Indicated power of engine	10·65 horse-power	... 12·40 horse-power
Water pumped per minute (measured)	80·38 gallons	... 90·08 gallons
Power in water pumped (including friction of pipes)	3·39 horse-power	... 4·00 horse-power
Useful effect	31·88 per cent.	... 32·25 per cent.
Engine, number of revolutions per minute	190	... 204·5

Observations.	Sept. 14, 1892. Average of Seven Tests.	Sept. 23, 1892. Average of Seven Tests
Dynamo, volts	408	456·3
„ ampères	14·1	14·2
„ output	7·74 horse-power	8·68 horse-power
Loss in engine and dynamo	27·73 per cent.	30·00 per cent.
Pump, revolutions per minute	42	47·43
Pressure of rising pipes	50·5 pounds	53·20 pounds
„ of suction-pipes	10 „	10·40 „
„ of pipes, pump standing ...	42 „	42 „
Friction of engine-belt and dynamo at 206 revolutions per minute, brushes on		4·17 horse-power
Friction of engine-belt and dynamo at 200 revolutions per minute, brushes taken off		3·32 „
Friction of engine alone, without the belt		2·24 „
Per cent. of horse-power to drive above (engine, belts, and dynamo) ...		33·64
Pumps' displacement per minute ...	97·17 gallons	109·73 gallons
Slip in pump	17·27 per cent.	17·90 per cent.
Motor, revolutions per minute... ...	1,044	1,180
Dynamo, revolutions calculated from engine revolutions		859
Rising main head due to pressure ...		95 ft. standing
Suction-vacuum head due to suck of 10·40 feet		24 ft. running

Mr. G. B. Forster asked if the water was measured?

Mr. Leach said it was, and in both tests the observations were continued simultaneously every tenth minute for an hour, and the average of the seven taken. The pressure of the water against the rams was taken by an indicator, screwed into the rising main, and the suck or pull of the pump by screwing it into the suction-pipes, and pulling the indicator barrel round by hand slowly when pressing the pencil against the card.

Mr. A. L. Steavenson (Durham) said a reference to Mr. Palmer's paper* gave his statement that "rightly or wrongly, from the experience acquired in experimenting with this plant, the writer is strongly of the opinion that, owing to the extremely diversified conditions which are met with in the various electric pumping arrangements, particulars of which are before the public, it is quite impossible to base any reliable conclusions upon the so-called plant-efficiency of the various installations." But they had never heard of this when talking of compressed air or ropes. It was only when electrical plant was brought into consideration that they were told it was impossible to arrive at real results. He (Mr. Steavenson) would like to draw attention to page 286,† where Mr. Selby Bigge said

* *Trans. Fed. Inst.*, vol. iii., page 276. † *Ibid.*, vol. iii.

"the calculation therefore of the power wasted in transmitting power by means of electricity is extremely simple, and the efficiencies of a plant can be readily foretold," etc. Mr. Selby Bigge did not appear to agree with Mr. Palmer, and where doctors differed others might take their own view of the matter. His own view was that if the same principles were applied in obtaining the useful effect, that compressed air would compare favourably with electricity. Mr. Palmer wished to take the brake-power—the power simply after it passed out of the motor! Decidedly not! because they were there losing the very essence of the electric arrangement, and that essence was, if they had a motor running at 1,100 or 1,200 revolutions per minute and pumps only at 15 or 20 revolutions they could not reduce the speed of the motor from 1,200 to the pump at 20, without losing a large amount of power. He had tried with all sorts of engines, and he knew that gearing which was a necessary part of electrical work was very destructive of power, and it was here that the chief loss occurred. In one of Mr. Palmer's plants he reduced his power 42⅔ times. This was where the loss came in. He did not mean to say they could not transmit the power between the dynamo and the motor with only a small loss; he would grant 90 or 95 per cent. of efficiency. The power was in the motor as the Irishman's purse was in the sea. On the same principle he supposed they would contend it was not lost because they knew where it was, but still they could not use it. He understood Mr. Leach's practical results amounted to this: That there was nothing very encouraging in using electricity as compared with other motive powers. Had he (Mr. Steavenson) been in Mr. Palmer's place and been asked to put down a plant under the same conditions he would no doubt have adopted electricity, as the distance was very great, and the cables could be moved about very readily. If it were not for the danger of electricity in the face, where they had to use safety-lamps, it would, he thought, be the very best motive power there; but where safety-lamps were necessary it required careful consideration as to whether that was a safe place for a motor. There was one thing in favour of compressed air: it was very pleasant and beneficial in the close atmosphere of a mine; they should not forget this desirable feature when considering the advantages of compressed air.

Mr. Thos. E. Forster (Newcastle-upon-Tyne) said he had had a short experience of electric pumping-plant at North Seaton colliery. It had not been running long enough to make tests such as those submitted by Mr. Leach, but it had otherwise given every satisfaction. The pump was delivering about 250 gallons of water per minute; the distance from the dynamo to the motor was about 2,700 yards, including the depth of

the shaft, as the dynamo was placed on the surface. The water was lifted about 50 feet, and passed through about 1,300 yards of 8 inches pipes. In connexion with this installation there was perhaps only one innovation, and that was in regard to laying the concentric cables. These had been placed underground, except in one or two places where the ground was wet, where they were enclosed in wooden boxes. He thought this arrangement rendered the cable safer, especially in cases like theirs, in which the cables had to be taken down a heavily timbered road, driven through old pillars. It was almost impossible to prevent the wires being disturbed where there were many repairs to be done. He quite agreed with Mr. Palmer's remarks as to the difficulty of recording the efficiency of these experiments, but according to Mr. Steavenson's remarks it was not fair to compare the results with those of compressed air unless they had some direct means of comparison. He would be very glad to communicate to the Institute the results of any tests which they might make at North Seaton colliery.

Mr. T. O. Robson (Gateshead-upon-Tyne) thought there might be a little misapprehension, but he understood Mr. Forster to say that burying the cable and taking it through boxes where the strata were wet would do away to a large extent with the necessity for safety-cables.

Mr. T. E. Forster said that where the cables were not buried they were temporarily raised and placed in boxes to keep them dry. Where they were buried there could be no danger arising from falls of stone, etc. Of course the cables were waterproofed, but it was thought best to keep them dry.

The President remarked that the cables would, of course, be perfectly safe as long as they were not broken.

Mr. T. E. Forster replied that he did not think they could easily be broken when placed underground.

Mr. G. S. Corlett (Wigan) said that as he had been responsible for the electrical portion of the plant described by Mr. Palmer he would like to make a few observations on it. With reference to the resistance of the old wire-rope, Mr. Walker was right, although for economy it was tempting to use old wire-ropes. The consideration of economy in first cost induced them to use wire-rope at East Howle colliery. They had had considerable difficulty with the connexions, and some portions of the wire-rope itself offered a higher electrical resistance than others, so that there was a consequent amount of loss. The electric plant was put down as the cheapest and speediest way of dealing with the water at the colliery and not with the view of publishing interesting results. High

efficiency was about the last thing considered. The steam was obtained from boilers heated by the waste gases of the coke ovens, so that a few horse-power of steam wasted was of no great moment, especially as the pumping was done at night, when there was a large quantity of spare steam. The only question considered was to keep the cost down as low as possible. He was not at all sure that Mr. Walker was right in saying that this plant had the lowest efficiency on record. The term "plant-efficiency" was a very misleading one, and it would hardly be fair to compare the plant-efficiency at East Howle colliery (from the plant-efficiency point of view) with the results that might be obtained with a plant like that Mr. Leach referred to. He claimed, however, that they were doing more work at East Howle colliery, although the theoretical power in the water itself might not show it; he contended that they were dealing with a larger quantity of water than had been attempted elsewhere in this country, and, taking into consideration the efficiency of the transmission, they were getting very satisfactory results. The plant was originally laid out to deal with 500 or 600 gallons of water per minute; now they got 900 gallons, and shortly the plant would be dealing with 1,600 gallons per minute. With regard to the loss of power in gearing, results differed considerably, but he thought, if the gearing were properly designed, this was not a serious matter. In a small pumping set which was put to work the previous week, he had found by careful measurement that the power absorbed by the motor when driving the pump at its normal speed was only 0·3 electric horse-power; the pump was a three-throw one, with $3\frac{1}{2}$ inches rams, making 30 revolutions per minute, driven by a Crompton electric motor running at 1,200 revolutions per minute. The pump and motor are fixed on the same bed-plate, connected by two sets of double helical gearing. He contended that the loss of only 0·3 electric horse-power in the motor, gearing, and pump friction was a very satisfactory result. Mr. Forster's plant might possibly be the largest pump in the kingdom, but it was only pumping 250 gallons per minute as compared with 900 gallons per minute at East Howle colliery. He would like to know, with so large a pump as that at North Seaton colliery, what was the loss in running the pump when empty? He might remark that in the electric pumping and hauling-plants working in South Wales it was the common custom to use a very heavily armoured cable and to bury it in about 6 inches of dirt.

Mr. A. L. Steavenson suggested that where electricity was adopted as a means of transmitting power they should adopt the centrifugal pump, and avoid the use of gearing when the height of column permitted.

Mr. Leach said electricians always raised the question of the size of the water-pipe; this had nothing to do with the efficiency or useful effect. With regard to the use of centrifugal pumps he had asked tenders for one to assist a syphon which had been troublesome.

Mr. W. C. Mountain (Newcastle-upon-Tyne) said he thought that the North Seaton pumping-plant was the largest in the kingdom, so far as the pumps were concerned, the rams being 9 inches in diameter and 15 inches stroke. The question of reducing the speed of the motor to that of the pump—from 720 revolutions to 30—was a serious one to deal with. There were several ways of doing it: the first by ordinary spur-gearing; next by worm-gearing; and thirdly by belts and other combinations. Worm-gearing was adopted for the North Seaton plant owing to want of space. The worm-wheel was made of phosphor-bronze, and the gearing perfectly true. It was pumping over 250 gallons of water per minute against a head of 50 feet, and the dynamo produced approximately 10 electrical horse-power. He would like to know how Mr. Palmer reduced the speed, and how Mr. Selby Bigge found spur-gearing to work at the high speed at which it had been running? With regard to the cables at North Seaton colliery they were placed underground at his suggestion. He had seen several of the so-called safety-cables, and if any of the members had any experience of these he would like to know how they behaved in practice. His own impression was that they could not avoid a certain amount of sparking, and probably they would not always act at the right moment. It seemed a pity to use a cheap cable simply because it reduced the cost; in a case of this kind people should be disposed to spend a little money in the first instance to put down a thoroughly reliable plant. In regard to the efficiency of the North Seaton electric pumps it was only fair to say that in a small engine such as Mr. Leach used the efficiency could not be very high, and it would be better to take the electrical efficiency of the plant itself.

Mr. Leach said he was not able to give the power going into the motor, as they had not a voltmeter. The 2·24 horse-power used to drive the engine alone amounted to 18·06 per cent., and although that was rather high it would be equally so with compressed air or any other plant. They could not discard the engine, seeing it was the motor power whether using electricity or air or anything else.

Mr. M. Walton Brown asked whether the use of safety-cables was necessary? If they were efficient there were still other parts of the plant which might become heated, such as the resistance-frames, etc. If Mr.

Mountain or Mr. Forster had made any test, he would like to know whether the insulation of the cable had been maintained as good as when the plant was erected?

Mr. Mountain said the cables at North Seaton colliery were concentric, insulated with bituminized fibre, and finally drawn into lead piping. They were then braided outside and covered with bitumen, so that it was almost impossible for the wet to destroy the insulation.

Mr. T. E. Forster said that at North Seaton colliery the cables were only buried where the ground was dry.

Mr. Mountain said the cable had only been laid about three months, and it was hardly fair to take a resistance test at present.

Mr. Walton Brown suggested that the Atkinson cable at Newbottle collieries might be tested by trying the effect of breaking the cable in the engine-house at bank when the full load was on.

Mr. Leach remarked that each switch was in a gas-tight case.

Mr. W. C. Blackett (Durham) said there had been no cases brought before the Institute in which the so-called safety and other cables had been tested in a practical way. In answer to a question he had put at a previous meeting one gentleman admitted that non-sparking was not in accordance with theory, and the claim that they did not spark in use was based upon laboratory experiment only. He (Mr. Blackett) was not an electrician, but looking at the matter from a common-sense point of view he failed to see how they could break a wire without getting a spark. As regards the lead-covered cable he would have liked to hear the result of actual experience. He had a small experience—scarcely worth mentioning—in conjunction with the more important plants described. In lighting a pit 100 fathoms deep, he had been obliged to use a bitumen-covered cable after the failure of two lead-covered ones. The first lead-covered cable failed in a very short time; the pit was wet, the lead seemed to be eaten away at once, and the current leaked into the brickwork of the shaft. The makers said they could supply a better lead-covered cable, and it was tried, but the result was the same as before.

Mr. T. E. Forster asked if the cables were covered with anything besides lead?

Mr. Blackett replied that the first was simply lead-covered, the second had a bituminized cover over the lead. Insulators were used, but a distinct current could be felt by placing the hand on the brickwork in the shaft.

Mr. Mountain asked if the cables themselves were thoroughly insulated?

Mr. BLACKETT replied that they were bought as such.

Mr. MOUNTAIN—What was the insulation ?

Mr. BLACKETT did not know.

Mr. MOUNTAIN could not understand if the pit water was pure water how it could affect the lead covering ; of course a chemical action in the water might be the real cause of the mischief.

Mr. STRATTON thought the fact of feeling the shock through the brickwork proved defective insulation.

Mr. BLACKETT said of course that was what he complained of.

Mr. STRATTON meant that the cable might not have been carried down the shaft on proper insulators.

Mr. BLACKETT said the cable was insulated, but it was thought that the wet state of the shaft destroyed the effects of the insulators.

Mr. T. E. FORSTER suggested that a test of the water in the shaft might reveal something.

Mr. BLACKETT said the water had been analysed, but nothing of an injurious nature was discovered in it.

Mr. A. L. STEAVENSON said he understood it had been suggested that the loss of power at the engine should not be reckoned, but it was so when other systems were tested.

Mr. MOUNTAIN explained that his contention had been that as the efficiency of engines varied considerably—that of an engine running at 100 or 200 revolutions per minute being much higher than one running at 10 or 15 revolutions—this loss should not be debited to the electrical plant.

Mr. HENRY PALMER, replying to the discussion, said he agreed in the main with Mr. S. F. Walker's remarks as to the use of old wire-ropes; but, as Mr. Corlett had pointed out, the only reason for using wire-ropes was on the score of economy in first cost. He would say plainly that if he were again required to put down an electric plant, after the experience he had gained, he would not hesitate to put down a double insulated conductor. As he pointed out on page 273* the wire-rope was supplemented by copper wires in order to reduce the loss, which amounted to something like 200 volts out of 550. He did not think he had advocated the use of old wire-rope, for they had found it very unsatisfactory. Mr. Walker also referred to the small water-pipes, but as had been explained in the paper these were already in position, and rather than face an expenditure of about £800, they elected to put up with a certain loss of efficiency of which they were fully aware. With reference to Mr.

* *Trans. Fed. Inst.*, vol. iii.

Steavenson's remark as to plant-efficiency some little mistake seemed to have crept in ; what he meant to convey was that nearly every plant differed from every other owing to local circumstances, and it seemed to him that for the purposes of comparison it was better to stop short at the work delivered on the pulley of the motor and ignore the work done in friction of water lifted, in comparing the efficiencies of electric plant. He gave the actual comparison of the work done in the water lifted compared with the indicated power of the engine, and the result to most people appeared low—16½ per cent. (Mr. Walker said the lowest on record.) He was not ashamed of the result, for it answered the purpose very well. The plant saved the pit from absolute drowning on three separate occasions, and they had every reason to be satisfied with the result. The plant-efficiency could have been made greater if they had gone to a higher first cost. He only contended for his own case ; Mr. Steavenson seemed to think he proposed to extend it to other cases. It was hardly fair to compare the East Howle efficiency of 16·4 per cent. with that of North Seaton of 46 per cent. They should consider that in one case they were pumping 900 gallons through a 9 inches water-pipe, and in the other only 140 gallons through a 6 inches pipe. The question of gearing was a mechanical one, and there was a good deal in Mr. Steavenson's contention that they could not reduce from 1,100 to 40 revolutions without some loss in the gearing. This loss was, unfortunately, unavoidable in the present stage of their experience, and until they could devise some other arrangement of running motors at a lower speed or running plants coupled up direct, they would have to put up with it. No doubt in a few years' time they would have more knowledge and be able to put up more economical plant than at present. He had no knowledge of the use of electricity at the face ; at present he had only put on record what they had done where employing naked lights, and until they had something more before them on the subject he would not care to go beyond this. He thought Mr. Mountain suggested the elimination of the engine, and starting with the power delivered into the dynamo at bank, and utilized in the water below. It did not much matter, however, which way it was taken, so long as it was understood ; but the same might be urged in regard to the experiments made in past years on fans. He was glad to hear what Mr. Blackett said as to the lead-covered cable. It occurred to him that, say with buried cables on engine-planes, it was a frequent practice to take ballast and such like material to make the roadway, and if water accumulated this might produce an undesirable liquid to act upon the cable—something of the nature of sulphuric acid, which would speedily effect a great deal of harm. With reference to Mr.

Stratton's remarks on insulated wires, no matter how they insulated with the earthenware insulator, if the shaft was wet they would get leakage; he found this at once with their old ropes when he stood in a damp place. In conclusion, he now considered that the results of the test referred to on page 277 of his paper* should be summarized as follows:—

	Indicated Horse-power.	Percentage of Work.
Loss in engine	12·00	22·33
„ generator	3·62	6·73
„ cables	7·04	13·09
„ motors	3·36	6·25
„ pumps	5·34	9·93
„ pipe friction	13·60	25·30
Water lifted	8·80	16·37
	53·76	100·00

The two items of pipe friction and water lifted give, when added together, the useful effect of the plant, which was equal to 41·67 per cent. The details of the calculations of the pipe friction are:—

Length of Pipes. Yards.	Diameter of Pipes. Inches.	Water Pumped. Gallons.	Head of Water. Feet.
750 ...	9 ...	delivery, 880 =	40·4775
50 ...	9 ...	„ 440 =	·6745
50 ...	9 ...	suction, 440 =	·6745
132 ...	9 ...	„ 340 =	1·0626
200 ...	4 ...	„ 100 =	8·0200
	Total		50·9091

Or 51 feet of head due to pipe friction, equal to 13·60 horse-power.

Mr. D. Selby Bigge wrote in reply, referring to Mr. Leach's remarks and the particulars given of the tests which resulted in a useful effect of 32 per cent. being obtained with the Seghill plant, that the engine for driving the electric pumping plant was about three times larger than necessary, provision being made for driving an electric lighting dynamo as well. With a small head and large suction-pipes the slip in the pump would be very materially increased; and, taking the small size of the plant and other things into consideration, the result is very reasonable. With reference to Mr. A. L. Steavenson's remarks, the writer maintains that, provided all the conditions are given, the exact useful effect which will be obtained can be computed far more accurately in the case of electricity than for compressed air or any other means. It has been the writer's experience that one can predict with almost absolute exactitude the current required by a machine in order to do a given amount of work, the current being conveyed through any given length of cable of a

* *Trans. Fed. Inst.*, vol. iii.

section previously determined. Makers of electric power plants are prepared to guarantee within 5 per cent. the exact results they will obtain, a guarantee that makers of compressed air plants are not prepared to give. The loss in gearing should not exceed 10 per cent. if the gearing is accurately made and fitted. The writer would be very glad to hear what are the results which have been obtained by the electric pumping plant at North Seaton colliery, of which a few particulars were given by Mr. T. E. Forster. He does not think, however, that a very high efficiency will be obtained, as the head is small and the quantity of water pumped at present not very considerable; and he believes that the voltage adopted was low, not exceeding 300 volts. The writer can point out that no case of injury to persons has been recorded with pressures of 800 volts downwards; indeed, he can personally testify that he has received shocks of this pressure. The very obvious advantages derived from a small cable, the small currents to be handled, the great reduction in first cost, and the efficiency which is obtained, point to the desirability of using the highest possible voltage consistent with the safety of the workpeople employed. All cables are liable to damage from various causes; and the writer's view is, that that cable which is the most accessible to examination and repair is the best for use in mines, and a well insulated cable run on insulators is to be preferred to a buried cable. Mr. A. L. Steavenson suggests the use of centrifugal pumps in order to avoid the use of gearing: they are very suitable for small heads and large volumes of water, but electrical engineers are especially concerned in cases where considerable distances are involved and small volumes of water to be lifted great heights. Mr. Leach alleged that the size of the water-pipe had little effect upon the efficiency of the plant. The writer has found, by practical experience, that with heads of from 100 to 400 feet the pipe friction formed from 100 to 25 per cent. of the total work, and that the average friction in colliery pipes was from three to five times as great as that given by the highest of recognized rules and tables. The writer's firm had found that, except in the case of very low heads, worm-gearing was inadmissible owing to the great loss of power by friction, particularly when the worm-gearing became somewhat worn. He agreed with Mr. Mountain in saying it would be better to take the electrical efficiency of the plant at Seghill, owing to the smallness of the motor engine. The writer, however, hopes shortly to lay before the members of the Institute a series of complete tests made on electric power plants varying in size from 5 to 300 horse-power.

The PRESIDENT thought the discussion on these papers must now be

closed, though he felt sure they would still hear much on the same subject. Electric transmission was in its infancy and it would be employed in many places where it was impossible to convey other power. It was a great addition to the power at the disposal of the mining engineer, and as time went on, and they got to know more about it, its efficiency would no doubt be much improved.

DISCUSSION UPON MR. J. L. SHAW'S PAPER ON "THE HÆMATITE ORES OF CUMBERLAND."

Mr. J. M. Main wrote that he had had over twenty years' experience in the working of iron ore in Cumberland, and had taken considerable interest in the various theories which had from time to time been propounded as to the mode of hæmatite deposition. With reference to this question, he would mention one or two prominent facts which had come under his observation, and which were altogether antagonistic to the explanation given by Mr. Shaw. Although there was much in the paper open to criticism, he would confine himself at present to the remarks bearing directly on the mode of deposition. In dealing with this part of the subject, Mr. Shaw mentioned that it was a simple matter to account for the support of the roofs of the supposed caverns before they were filled with iron ore. His remarks on this point were certainly curt and vague. He says* "It is known that there are many points of support, in addition to the rock at the outer run or edge of the deposit, where the roof stone meets the sole without any intervening ore. One of these points is shown in the section (Fig. 3, Plate XXXVII.) across the Hodbarrow mines. The overlying limestone must have been steadied in this way, and if at any point the limestone was not supported, and fell upon the sole before the deposition of the ore took place, then at that point the ore would not be deposited. The above remarks especially apply to the iron ore mines in the Egremont district." An explanation of this kind, to be consistent, must apply not merely to a particular part of a district but to the whole field. The writer is acquainted with several large deposits scattered over the hæmatite-bearing measures of Cumberland, where the roof of the ore-bed, shelving off from faults, stretched at right angles from these faults to a distance of from 50 to 150 feet, the roof in some cases being limestone, with "pockets" of shale extending upwards for several feet, the limestone itself not being of that character which would support a superincumbent weight anything like so great as would prevail during the time

* *Trans. Fed. Inst.*, vol. iii., page 589.

the pre-existent caverns, described by Mr. Shaw, remained unfilled. In other cases the roof consisted of a shale-bed of a soft and brittle nature which could not possibly support its own weight, even if the space were only a tenth part of that indicated. These shale roofs are interbedded with the surrounding limestone. As an instance of the latter, the writer may mention the deposit of iron ore in the Crossfield Company's mines, extending from No. 7 pit westward to the main north-and-south fault, running through that portion of their estate; the lateral spread of the ore-bed was about 75 feet, while its longitudinal extension, embracing the above width, was 200 feet. The average thickness of the ore was about 9 feet, and that of the overlying shale 6 feet. Here and there, blocks of siliceous limestone occurred above the ore, but under the shale. These blocks when undermined had to be securely propped to prevent them falling. No stretch of the imagination could reconcile either of the above conditions with that part of Mr. Shaw's explanation contained in the paragraph above quoted. Another point, which seemingly had escaped Mr. Shaw's observation, but which had been prominently brought forward by Mr. J. D. Kendall in his paper on the "Hæmatite Deposits of West Cumberland," and which unquestionably was a clenching argument in favour of the displacement theory, was the fact of shale-beds extending not only across an ore deposit but into the limestone which formed the surrounding walls. Mr. Kendall, in his paper on the "Hæmatite Deposits of West Cumberland," gave a section showing the position of such a bed in the Crossfield Company's No. 8 opencast workings. The writer can testify to the accuracy of this section, as in several places the bed was followed for a considerable distance into the hanging wall. The ore at this part was from 20 or 30 feet from wall to wall, whilst its thickness both above and below the shale-bed was also considerable. This shale parting or bed must have existed before the ore was deposited, and must have remained suspended in empty space during the time Mr. Shaw's presupposed caverns were being filled with iron ore, by aqueous agency, from the bottom upwards, which is an utter impossibility. This fact was a complete proof that the hæmatite deposits in Cumberland were formed by replacement, as first suggested by Mr. Kendall, and demonstrated frequently by him with an array of facts and careful deductions, which, for completeness, have not been approached by any other writer. He (Mr. Main) had found a number of the fossils, common to the district limestone in the ore itself, the presence of which in such a position could not be explained in any other way than that advocated by Mr. Kendall.

Mr. J. D. KENDALL wrote in reply to the remarks of Mr. J. L. Shaw* on four points: firstly, as to the thin shales that are interbedded with the hæmatite and the surrounding limestone; secondly, as to the nature of the Gillfoot Park deposit; thirdly, as to the so-called limestone-bed in the breccia; and, fourthly, as to the quantity of calcareous matter contained in the thin ore-bed at Bigrigg. The writer put a number of questions to Mr. Shaw which that gentleman has not attempted to answer, nor shown that they were irrelevant to the points at issue. Surely it was incumbent upon him to do either one or other of these things, or to admit frankly that he could not. Even the latter course would have been much more satisfactory to members than a regalement of storyettes *apropos* of nothing. The writer did not, however, intend to let Mr. Shaw off in this easy manner, but again invited him to pursue one or other of the courses just indicated. If the explanation of that gentleman was worth anything at all surely it was worth defending and making intelligible. But to come to the particular questions that induced this communication, Mr. Shaw said,† he had been "employed twenty-two years in the iron-ore mines of Furness and Cumberland without seeing a bed or parting of shale running through a gut of ore and into the limestone on either side." This is not surprising when we remember how much people look at and never see, but in Mr. Shaw's case it was only what might be expected when it is known that his experience of the deposits mentioned by the writer as containing these interbedded shales is absolutely *nil*. A man might spend a hundred years in different ore deposits of this district and yet never see one of these shale partings, for the simple reason that there are only some deposits which contain them. They are never found in deposits in the first or second limestone of Cumberland, and very rarely indeed anywhere in Furness. But if Mr. Shaw's cavernous and sedimentary explanation be correct why should not these partings be found in all deposits alike? On the other hand if the deposits originated by replacement it would be useless looking for interbedded shales in deposits of the first and second limestone of Cumberland, because such shales do not exist in the limestone, nor are there many in the thick limestone of Furness. It is difficult, however, to rid one's self of the impression that Mr. Shaw had not been very anxious to find these shale partings, or it would have been a very easy matter for him to have asked the writer where any could be seen, which he never did. And now as to the nature of the Gillfoot Park deposit. Mr. Shaw says he "does not think that Mr. Kendall has had any previous

* *Trans. Fed. Inst.*, vol. iii., pages 601-603. † *Ibid.*, vol. iii., page 602.

acquaintance with this deposit, because he speaks only about the eastern side of the bed-like deposit being tilted, and takes no notice of the great downthrow fault on which the vein-like ore lies 900 feet to the west. Mr. Kendall is clearly in error, his sketch incorrect, and his idea of the beds being tilted is quite erroneous." In reply to the first part of this statement the writer would like to say that he is not in the habit of writing on subjects of which he has not had experience, and that not only has he had experience of the Gillfoot Park mines and the Whin Pit under the cemetery, but of every other deposit in the Whitehaven district and Furness. This is something doubtless outside Mr. Shaw's knowledge, but the writer hoped that that gentleman will not therefore treat it as apocryphal as he does the shale partings, simply because he knows nothing about them. Mr. Shaw surely cannot see the full force of the implication contained in his statement. Moreover, it is impossible to argue with anyone who assumes such a position as he had done. If he were to treat as apocryphal everything that he had not seen almost the entire universe would be to him but a fiction. As to the accuracy of the section of the rise part of Gillfoot Park mine, given by the writer, he can only at present re-affirm it, knowing that it represents substantially the facts. It can, however, easily be verified or disproved by going down the Whin Pit shaft, which was still open, and so doubtless was the drift therefrom, which cut through the red shales underlying the ore. This would involve some cost, but the writer was quite willing to bear that should his section be proved to be wrong (*i.e.*, if the beds be not tilted as he says), provided Mr. Shaw would take upon himself a similar responsibility in case the writer's section, on independent evidence, was found to be right. This seemed to be the only mode of settling the difference between us, otherwise affirmation and denial might go on interminably. Mr. Shaw's remarks about No. 4 Pit at Gillfoot Park, sinking through the so-called limestone-bed, did not affect the matter in dispute. The writer had seen several pits sunk through these limy bands, and the sinkers were invariably of opinion that they were going through a broken limestone-bed. In reality it was only a very limy part of the breccia. Mr. Shaw was under a misapprehension in supposing that anyone had indicated that he was not a theorist. However, it was highly satisfactory to know that his mind was strictly practical, for he would be the better able to deal with the following very practical question. If Gillfoot Park deposit was formed in a cavern, will Mr. Shaw say why the lessees have not been able to extract the ore worked by pits Nos. 1 and 2, and leave the original cavern intact? As a matter of fact the whole roof has collapsed, and settled down upon the

floor. The surface of the ground subsiding correspondingly. We feel very sure that if we were to fill up, say the Peak Cavern or Kent's Hole, or any other of the numerous caverns in the country, we could practically exhaust them again without injuring the strength of the roof in any way. And why? Because the span of the roof in every case is so small that there is abundant strength. The great downthrow fault which Mr. Shaw says the writer takes no notice of is actually shown in his (the writer's) section. Again Mr. Shaw writes "there is nothing in Mr. Kendall's remarks about there being calcareous rock in the ore-bed at Bigrigg; that is common enough. There is much less lime than sand in its composition, the stratum being composed of hæmatite, sandstone, and calcareous mudstone." To this the writer would reply that he does not believe there is a grain of sand in the bed. There is some re-deposited silica, but not much, whereas there are large portions of the bed almost entirely calcareous. But to settle the point the writer is prepared to have an independent analysis made on the terms mentioned above with regard to the testing of the facts at Gillfoot Park. Lastly, Mr. Shaw says the "growing together of hæmatite and limestone is wrongly expressed, and conveys a wrong meaning," but he does not tell us how to express the fact, nor what is the wrong meaning which it conveys. Will he kindly do so? The writer's chief anxiety is about the facts, and therefore he would most readily express any of them differently if by so doing they became more intelligible. This leads us to another remark of Mr. Shaw's that "the different classes of matter (in the ore) became ultimately separated by the affinity of particles." What does this mean?

Mr. J. L. Shaw, in reply, wrote that since replying to Mr. Kendall's first remarks he had carefully looked over the sections contained in that gentleman's paper on "The Hæmatite Deposits of West Cumberland," which show nothing against the idea of the deposition of ore in caverns, and the aqueous and chemical formation of hæmatite. The writer has given his opinion in regard to the Gillfoot Park deposit that "in this case at all events the cavern was excavated first and the ore deposited therein afterwards." Mr. Kendall assumes when he mentions a bed of shale running through a body of ore, and into the limestone on one or both sides, that it is necessary that the whole of the limestone which the ore has replaced should first have been carried away, according to the writer's theory above-named. But the writer did not convey that meaning; and it was not necessary that he should, while still retaining his opinion as to the ore having been deposited in caverns, because in many, if not in all cases, the wearing action on the

limestone, and the deposition of iron may have gone on simultaneously. Time is a very great element in the work of nature, so that while the limestone was being softened and worn by acidulated water (beginning of course at the point next the fault or joint to which the water had readiest access), at the same time and from the same water iron was being deposited. This process would go on until any shale-bed was reached, when a new bed would already have been formed for the shale to rest on, and it was not necessary that the water should have washed away the shale-bed, at all events in all cases, but that the same process should have gone on until the whole cavern was filled. There is not in any of Mr. Kendall's sections a case to which this method of formation would not apply. The writer had never seen the necessity of bringing in shale-beds to support his opinion on the formation of hæmatite, and he did not therefore think it necessary to ask Mr. Kendall where the shale-beds above referred to were to be seen, but was guided by the sections in Mr. Kendall's paper. Mr. Kendall looks at the matter from an entirely different standpoint, and it was not necessary therefore that the writer should see the relevancy of some of his questions, and as for opening out the Whin Pit again the proposal was not practicable, for the level, from the shaft at all events, had doubtless entirely collapsed. The writer's idea in regard to Mr. Kendall's Gillfoot Park section was that he depended upon his imagination when it was drawn. It was not even drawn to scale like the sections connected with Mr. Kendall's own papers. The writer was not therefore assuming too much when the idea entered his mind that he (Mr. Kendall) had "no previous acquaintance" with the Gillfoot Park deposit. He could not deny that the beds of limestone and shale on the hanging side of the vein under the cemetery were steeper than the general run of the bedding to the west of this fault, but he understood Mr. Kendall, by his statement of "the beds being tilted at a high angle by the fault A" to mean that the lying side had been forced up while the writer's idea was that the hanging side was thrown down. It might be said that there was practically no difference, but attention must again be called to the fact that Mr. Kendall in his first remarks said that the writer was "in error" in regard to the Gillfoot Park deposit in general, and to this point in particular. The writer did not depend on the testimony of the sinkers in regard to the bed of limestone mentioned as having been sunk through in the No. 4 Pit at Gillfoot Park, but on that of Mr. James Stout, the manager, who was of the same opinion. Touching Mr. Kendall's "very practical question" "will Mr. Shaw tell us why the Gillfoot Park lessees have not been able to extract the ore worked by pits Nos. 1 and 2, and

leave the original cavern intact?" it was amusing to preceive Mr. Kendall's non-acquaintance with the Gillfoot Park mine, even though "he has had experience of that and every other deposit in the Whitehaven district and Furness." Mr. Kendall says his question is very practical, but the writer must point out that the conditions of mining by man and by nature, so to speak, are very different, and he does not think the members will expect him to enter into the details of mining at Gillfoot Park. The conditions being entirely different, it was quite wrong to make a comparison between the two, man had to make ways—travelling and other ways—which nature had not to do, and he must also point out that water could make its way through cavities, which man could not see with his naked eye. But as before remarked it was not necessary in order to controvert Mr. Kendall's "replacement theory" that the whole of the limestone should have been removed before the iron was deposited, but the two processes of wasting and deposition may have gone on simultaneously in all cases. Besides visiting many or most of the caverns in England Mr. Kendall has been in the United States. Has he visited "the Mammoth Cave of Kentucky?" Is not the area which there stands unsupported at least as great as the Gillfoot Park deposit, and has it not stood so for ages? It was not necessary to make an independent analysis of the thin stratum in the cutting of the mineral railway at Bigrigg, because the mixed nature of the stratum was easily seen, and he had taken specimens from the bed which consisted of hæmatite, mud and sand, or rather "what had once been mud and sand." "The growing together of the limestone and ore is left unnoticed." The writer differs entirely from Mr. Kendall on this point. As already explained he did not think that the limestone and ore had grown together" so much as that the latter had been "growing" while the former was being wasted by the ordinary operation of water. The writer had seen ore of the very best quality entirely surrounded by the cleanest limestone. One case was in the first iron-ore pit which was sunk at Stank-in-Furness just before the large northerly gut of ore was cut. A patch of ore was seen by the writer in the side of the shaft about 12 inches in diameter, and surrounded as above stated. It was surely absurd to ask "if Mr. Shaw's cavernous and sedimentary explanation be correct, why should not these partings be found in all deposits alike?" Neither Mr. Kendall nor the writer have the slightest doubt about the shales which sometimes part the limestone-beds being found in particular parts only, and that hæmatite was found in all parts of the limestone from top to bottom. The latter was an axiom of the profession generally, but these shales being in some cases interbedded, so to speak, with ore were not

at all essential to explain the formation of hæmatite. It was only Mr. Kendall and Mr. Main who look on these shale-beds as "a clenching argument" in favour of the former gentleman's "replacement theory." With regard to the writer's remark that "the different classes of matter (in the ore) became ultimately separated by the affinity of particles" he would answer Mr. Kendall's question by asking him another. How does he account for flints in chalk, otherwise than as the aggregations of particles from mutual attractions among each other as noticed in De la Beche's *Geological Researches*, page 94, when treating of the nodules in the Lias of Lyme Regis consisting of carbonate of lime in clay? On page 97 the same author says:—"Layers of nodules . . . . composed of substances which have separated out from the constituent parts of a mechanical rock after deposition are common in many strata. The ironstone nodules of the Coal-measures seem to have been thus produced." The writer had in his paper applied this principle to the clay and quicksand overlying the ore at the Hodbarrow mines. By the term therefore "affinity of particles" was meant the same thing as by "mechanical attraction of particles" in the part of his paper which referred to these clays and quicksands, the mud and sand in these diluvial drifts having doubtless been thrown down indiscriminately. In the case of the hæmatite deposits there doubtless was a difference in the rate of supply of material, atmospheric changes being sufficient to account for these differences, and for differences in the quality or composition of the ore. Mr. Main's reference to one of the writer's sections applied to Fig. 2, Plate XXXVII. (Gillfoot Park), and not to Fig. 3 (Hodbarrow mines). The writer had not the slightest doubt about the accuracy of Mr. Kendall's sections which were published in connexion with his own papers, and that the beds of shale shown on these existed before the ore was deposited, but he entirely differed from Mr. Main in thinking it impossible that ore should be deposited both below and above such shale-beds by the ordinary operations of water, and he could see no proof of Mr. Kendall's pet "replacement theory" in it.

Mr. Henry White then read his paper on "The Coad Electric Miner's Lamp" as follows:—

THE COAD ELECTRIC MINER'S LAMP.

By HENRY WHITE.

During the last few weeks the writer has several times tried the Coad electric safety-lamps underground, at the Walker colliery, and being highly pleased with the improved light thought that a description of the lamp might be sufficiently interesting to lay before the members of this Institute.

The No. 1 electric lamp gives a light much superior to that of an ordinary safety-lamp, and no doubt would enable work to be done more efficiently underground.

The No. 3 electric lamp should prove very useful for an official, in travelling and examining high places, timber, or loftings; also the rails, joints, lead, gradient, etc., of engine-planes and inclines and working-places generally. It might also be very usefully applied at off-takes, landings, bank-heads, stables, etc.; and at the bottom and top of staples where coals are drawn and safety-lamps are now in use. The writer considers it much superior to the ordinary safety-lamp for use in examining ropes, pulleys, stone, etc., in upcast shafts, staples, etc.

Experimentally it was found that with the ordinary tin-can Davy lamp a piece of white paper 6 inches square could only be seen about 10 yards off, whilst with the No. 3 electric lamp it could be seen at a distance of 40 yards.

This electric lamp reduces the use and generation of electricity down to the level of an ordinary oil lamp, and the light is given at considerably less cost, as one candle-power is given for 10 hours at the cost of one penny.

The battery is a primary single-fluid one, made of ebonite cells, with platinum connexions, which do not corrode in use, the usual fault and trouble with most batteries. No scientific knowledge of electricity is required in its use. To charge the battery the cover is removed, and the cells filled with the fluid; then the zinc plates are dropped in their places when they immediately connect and the cover put on again.

The small incandescent lamp is held on a flexible or yielding support which allows the lamp to be pushed on one side, to prevent it being broken on meeting with a blow; immediately this takes place the circuit of the current is broken, and the lamp goes out; the sparking caused by this

and the making contact takes place in a sealed chamber, into which no gas can enter. When any pressure that pushes the lamp on one side is removed the lamp comes back to its normal position, and is at once automatically relighted. The lamp is further protected by a stout glass envelope or cover.

Fig. 1, Plate X., is a side view of a three-cells miner's lamp. Fig. 2 shows the incandescent lamp pushed on one side with the outer glass envelope broken; in this position the incandescent lamp is extinguished. Fig. 3 shows a four-cell lamp, fitted with a powerful deflecting search-light or lantern, capable of throwing a focussed light of over fifty candle-power to a distance of many yards. This lamp can be detached (Fig. 4) from the battery and held in the hand for searching purposes, to obviate the necessity of turning or of holding the battery in inconvenient positions. Fig. 5 is a sectional view of the battery, showing the insertion of a zinc plate. This plate is supported on the projections A, one side of which rests upon the platinum contact-points B, thereby completing the connexion of the zinc with the carbon.

The weight of the lamp shown in Fig. 1, when fully charged, is 4 lbs. 2 ozs., and that of the lamp shown in Fig. 3, 5 lbs. 6 ozs.

One charge of 10 ounces of the improved bichromate-of-potash fluid will give ten hours' light.

The consumption of zinc is: for the three-cell lamp, $\frac{3}{4}$ ounce per ten hours; and for the four-cell, 1 ounce for the same time.

Mr. G. B. Forster asked if there was any apparatus connected with the lamp to indicate the presence of gas?

Mr. Coad (the inventor of the lamp) said there was not; he had not thought there was any necessity for it. Could not a man smell the gas?

Mr. Forster—Perhaps, but not in time to prevent suffocation.

Mr. W. C. Blackett asked if the makers of the lamp were prepared to maintain it at the cost named, of 1d. per candle-power per 10 hours?

Mr. Coad said they did not undertake this, it could be a matter for arrangement.

Mr. Blackett also asked if the fluid could be bought in the open market?

Mr. COAD said that arrangements could be made for colliery owners to make their own.

Mr. BLACKETT—Free of royalty?

Mr. COAD could not say that.

The PRESIDENT proposed a vote of thanks to Mr. White for bringing this electric lamp before their notice; it was very interesting and might be found very useful.

The motion was agreed to.

Mr. WHITE acknowledged the vote of thanks and said he had found the lamp very useful.

Mr. J. H. MERIVALE'S paper on "The Education of Mining Engineers"* and Mr. BRYAN DONKIN'S paper on "Steam Boilers with Forced Blast: the Perret System for Burning Dust and Rejected Fuels; with Notes on Testing Boilers" were taken as read, and the meeting closed.

* This paper will be published as an Appendix to the present volume of the *Transactions*.

STEAM BOILERS WITH FORCED BLAST: THE PERRET SYSTEM FOR BURNING DUST AND REJECTED FUELS; WITH NOTES ON TESTING BOILERS.

By BRYAN DONKIN, Jun., M.I.C.E.

Introduction.

In working modern steam boilers economically, the question is, not so much to burn any large coal obtainable in the market, as to consume, if possible, what cannot be burnt in the ordinary way. In other words, to go direct to the pit's mouth, or other centres, not to purchase good coal, but to find out and endeavour to use the refuse heaps that are often accumulating there, the cost of which, of course, is much less.

Any invention tending to promote the use of small or dust coal, coke, etc., is deserving the attention of steam and heat consumers, especially in colliery districts. Heat should be bought in the cheapest form and utilized in the best way for the production of steam.

A boiler is simply a heat apparatus to evaporate water under pressure by means of some kind of fuel. The object should be to evaporate the maximum weight of water per hour at the minimum cost of fuel. The products of combustion should not be allowed to escape too hot.

The best standard of comparison, as will be seen later on, is the cost of the fuel consumed per 1,000 gallons of water converted into steam. In this way boiler results can be readily compared in the same or different districts. The temperature of the water evaporated should be given, and also the pressure of the steam.

There is now a demand for furnaces to burn economically all kinds of refuse fuels, hard dust coke from coke-ovens, coke dust from gas-works, anthracite, washings of coal-mines, screenings, etc. Many grates are at work that will burn even house refuse from dust-bins, etc.

Use of Forced Blast.

It is impossible to burn those classes of fuels, which are in the form of dust, thoroughly, without mor .d more heat. In the ordinary grates the quantity of air ad . the grate is limited by the

pull of the chimney of say $\frac{1}{4}$, $\frac{1}{2}$, up to $\frac{3}{4}$ inch of water. The maximum vacuum the author has found in a very high chimney is about $\frac{3}{4}$ inch of water.

Hence the necessity for supplying large quantities of air under a small pressure to the furnaces. The easiest way of doing this is by means of a small fan placed near the boiler, forcing the required volume of air into the furnace under a water-pressure of about $\frac{1}{2}$ to $\frac{3}{4}$ inch. Provision must also be made for cooling the bars, to prevent the clinker adhering and the bars from getting hot, so that they may last several years. Of course, if a chimney exists, the vacuum or pull produced can be utilized as well as the air pressure under the bars. If a chimney has to be constructed, one of smaller height and much less cost than usual is sufficient with this system.

To ascertain, if required, whether too much or too little air is introduced, the gases of combustion should be analysed by means of a very simple apparatus.

It should be remembered that forced chimney draught has been used for more than half a century in locomotives, the exhaust steam from the cylinders escaping up the chimney causing a considerable vacuum in the chimney, the tubes, and the fire-box. In some locomotive experiments, made recently by the author, where the water and coals were carefully measured during a run of 100 miles, the vacuum in the chimney with 14 coaches going uphill was found, by the water-gauge, to be about $2\frac{1}{4}$ inches. On the level it was about 1 inch.

In modern marine practice, forced blast and forced draught have not been many years in use, but they are now very generally employed in tubular marine boilers either with or without closed stokeholes. The pressure varies from 1 to 3 inches, and is produced by fans.

In agricultural engines, as in locomotives, a vacuum is produced by the usual blast in the chimney, by means of exhaust steam.

The Perret System applied to Land Boilers.

This system was invented by the celebrated chemist and scientist, Mr. Michel Perret of Paris. His name is well-known in France as the inventor of several very ingenious kinds of furnaces for heating buildings, etc., and his arrangement can be applied practically to all kinds of steam boilers, either new or old.

Figs. 1, 2, 5, and 6, Plate XI., show the Perret grate as applied to the Lancashire boiler. It will be seen that the bars K are much thinner and deeper than is usual in ordinary grates, and present much larger

cooling-surfaces. On an average each bar is about $\frac{5}{8}$ inch thick at the coal-line, $\frac{3}{8}$ inch thick at the water-line, and about 2 feet long. The bars are placed very near together, being only about $\frac{1}{10}$ inch apart, and are 9 inches deep.

Under the grate there is a shallow, cast-iron, open tank or trough D, which is always kept automatically full of water, by means of a ball cock A. Into this tank, as will be seen in the figures, all the bars dip, so that they are immersed from $2\frac{1}{2}$ to 3 inches at their lower extremities. For this reason the Perret furnace is often called a "water-cased grate." The bars are kept much cooler than usual by their being constantly in water. The temperature of the water is generally about 110 degs. Fahr., and the evaporation from each trough about 1 gallon per hour. The vertical currents of air between the bars at atmospheric temperature also tend to cool them, as well as to prevent the dust from falling into the tank; the tank can easily be cleaned out every two or three weeks, through a special door G. One advantage of this system is that the bars are kept cool, and therefore the clinker does not adhere to them, as with ordinary bars. If the burning coals are removed from the bars, the bars can be touched by the fingers without danger. The clinker can, without any effort on the part of the attendant, be lifted and removed. For this reason these cast-iron bars last long and do not lose their shape in four or five years.

The temperature of the furnace is somewhat higher with forced blast than without; there is also better combustion, as the analyses of the chimney gases show, with the production of very little or no carbonic oxide gas. Another advantage is that a small quantity of the water from the open top of the tank, in which the bars are immersed, is continually evaporated. The moisture thus given off has been shown lately by experiments to assist combustion, and to diminish the quantity of clinker, as well as the smoke.

Figs. 5 and 6 show where the air enters by pipes E to each grate. The whole of the lower part of the grate, between the water-line in the trough and the coal-line on the top of the bars, is under pressure.

These furnaces are suitable for, and have been added to, all types of steam boilers, both with internal and external fires, burning all kinds of small and dust fuels. In factories and mills where there are many boilers, one boiler is sometimes fitted with this grate for consuming all the refuse and ashes coming from the other boilers.

The weight of fuel that can be burnt per square foot of grate per hour varies of course much, according to circumstances, but 10 to 20 lbs. is the

usual consumption. As much, however, as 30 to 35 lbs. have been burnt per hour per square foot. More even can be burnt, but this generally means waste of heat.

Figs. 3, 4, 7, 8, 9, and 10, Plate XI., show applications of this system to egg-ended, tubulous, and locomotive type boilers.

The economy obtained in practice by the adoption of the Perret forced-blast grate, as compared with the ordinary system, is generally about 50 per cent. in the annual coal bills, as much cheaper fuels can be burned, but it varies from 30 per cent. to 60 per cent. In London the economy is often 50 per cent., at collieries about 40 to 50 per cent.; it, however, depends, as in other places, upon the price of the dust fuel, as compared with that used previously. In France it almost always halves the coal bills.

With this system, if more steam is required for a short time it can be obtained. Generally boilers with the Perret grates can be made to burn more fuel and evaporate more water than with ordinary grates, but this depends on the quality of fuel. The blast can be regulated to suit the varying quantities of steam required during different times of the working day. With high or low class fuels it is not recommended to force boilers, as this means waste of heat and uneconomical working, unless large economizers are used to reduce the temperature of gases by putting some of this heat into the feed-water.

The bars are practically the only parts of the apparatus subject to wear, and these generally last four or five years.

The boiler flues in all boilers should be large so as to be readily cleaned. The brickwork should be good, and with an air space to diminish radiation. Fronts of steam boilers should always be covered to keep the boiler at as uniform a temperature as possible and to reduce the waste of heat.

On the Measurement of Feed-water to Steam Boilers.

To ascertain really what a boiler is doing, it is essential to know the quantity of water evaporated per hour for a given weight of fuel consumed. This is the only practical method of finding out the work done with a given quantity of combustible burnt or heat given off; this heat is taken up by a known area of heating surface exposed to the flames, and hot gases.

There is a great difficulty in getting mill owners, engineers, and others to measure their feed-water, and yet this is as easily done as measuring gas by meters. In collieries and large factories, with 10, 20, or 30 boilers, there is seldom a meter fixed even to one boiler. There are in England some 100 to 150 boilers provided with meters known to the author. He recommends that one boiler at least should be an experimental

one, to find out evaporative facts, and that it should be provided with means of measuring the feed-water, doing of course its usual work at the same time.

There are now many good serviceable water-meters that will register the feed-water, as it passes into a boiler from a feed-pump, within 2 or 3 per cent. of the actual quantity. The author has had a good deal of experience with different water-meters, and has tested many by means of gauged tanks. They should be lined with brass, and have brass piston rings. The piston type of water-meter is generally the best and most reliable.

A careful daily and weekly record of the weight of fuel burnt and the quantity of water evaporated always conduces to economy, as it enables comparisons to be made of different fuels, stoking, etc. A form recommended for this purpose is added. (See Appendix D.)

When all the water is used for power, the meter of course gives the consumption of the steam to the engine or engines, an important point to determine. At the Bermondsey works such a daily record has been kept for seven or eight years, and at many other factories the same is done.

The author considers that boiler owners and engineers should ask themselves the following questions, and make arrangements to solve all or most of them :—

1. What is the heating value of the fuel burnt ?
2. What is the evaporative power of the boiler used, having a given heating and grate-surface ?
3. What is the value of the economizer or feed-water heater ?
4. What is the rate of evaporation and combustion per square foot of heating and grate-surface ?
5. What is the best method of stoking and best thickness of fires ?
6. What is the temperature of the gases escaping up the chimney and wasted ?
7. What is the cost of the fuel used for the evaporation of 1,000 gallons of water per hour ?
8. What draught is the chimney giving ?

How few can answer these and other questions even approximately !

The number of pounds of steam produced for each pound of fuel should be known, or in other words, how much water is forced into the boiler by the feed-pump, to replace that evaporated into steam. A good plan, and one practised in France, is to give the stoker a pecuniary interest in the saving of fuel or in the evaporative results. Owners will soon recoup themselves for the cost of the meter, etc., by the economy obtained.

In one large mill in Belgium the owner pays the coal merchants, not by the number of tons of coal delivered and burnt, but by the number of

tons of water evaporated or steam made, so that the coal merchants are directly interested in sending him a clean, good, economical coal, with the smallest percentage of incombustible matter. This plan works well and seems an excellent one.

The meter should not be placed on the suction of the feed-pump, but on the delivery pipe between the pump and the boiler, and it should be provided with a bye-pass for the water.

Some thousands of meters are used in France, Germany, and Switzerland, both on sea and on land, for measuring feed-water.

Let us take a given district, say London or Newcastle, and let us suppose that it is desirable to compare the economic results of various fuels, and to have a practical standard of comparison for different boilers, fuels, and stokers, etc. This is the more important, as there are so many different classes of boilers in use, evaporating water with very different priced fuels, with different pressures of steam and different stokers, either mechanical or manual.

The cost of fuel per ton varies in London from 5s. to 24s., or in the proportion of nearly 5 to 1. In addition to this, the evaporation varies from 5 to 10 lbs. of water per lb. of fuel, or in the proportion of 2 to 1.

To compare these results, some standard should be fixed, combining the cost of fuel with its evaporative efficiency, or steam-producing power of the boiler. The best unit of comparison is, in the author's opinion, the cost of fuel for evaporating 1,000 gallons of water per hour at a certain temperature. In this way a boiler burning so many tons of coal, and evaporating so many thousands of gallons of water, can be compared with another boiler having a different evaporative power and supplied with a different fuel.

As steam boilers are required for evaporating water, the question is how to do it at the most economical rate. This can only be done by using a fuel giving the best evaporation at the price, which may be determined by some such method as here described. Three examples are now given to make the matter clearer.

Experiment	Kind of Fuel.	Cost per Ton.	Water Evaporated per lb. of Fuel by Experiment.	Cost of Fuel per 1,000 Gals. Water Evaporated.
			Pounds.	
A	Dust coke on Perret grate	5s.	6	3s. 8d.
B	Dust Welsh coal on Perret grate	10s.	$8\frac{1}{2}$	5s. 2d.
C	Ordinary large Welsh coal on ordinary grate ...	22s.	9	10s. 11d.

The above are all actual experimental results at the same steam pressure.

It will be seen that B is less than half the cost of C per 1,000 gallons of water evaporated, and A is considerably less than B.

The cost of using these three very different classes of fuels with the same boiler and the same stoker is thus in the proportion of the numbers 22, 31, and 65. Using dust Welsh coal in lieu of large Welsh coal will be seen to halve the coal bills for the same quantity of steam generated or water evaporated.

On the Belgian State railways, a proportion of all the coal supplied by the contractors is always tested under fixed locomotive boilers, and with the pressure of steam used on the railways. The coal is refused unless it passes the evaporative test of so many lbs. of water per lb. It is also rejected if the percentage of dirt or incombustible matter is above the limits defined. The same chimney blast is used as on the railways.

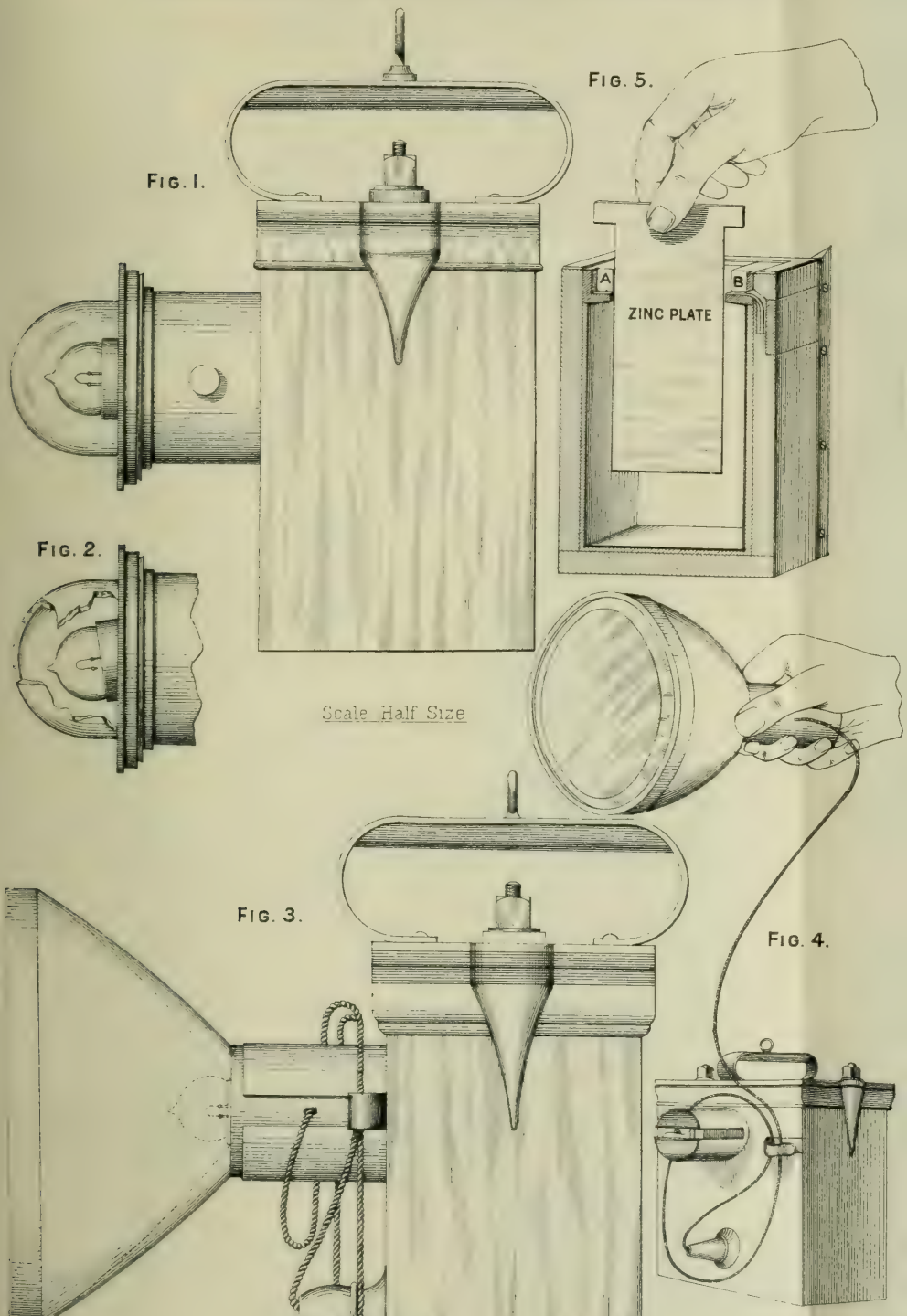
The author has assisted at one of these official tests, which are very well and carefully carried out by capable experts. He believes the same system is in force for the coal purchased by contract for the German Imperial Navy. In England we are too careless in these matters.

Many of the boiler associations, both in England and on the Continent, are continually doing excellent work in making and publishing accurate boiler tests in which the water, coal, clinker, etc., are all carefully weighed, and various other additional details are given. The gases of combustion are analysed, and their temperature noted, both before and after passing the economizer.

If coal users are buying an ordinary coal, they often pay 1s. to 2s. per ton for useless incombustible matter. The weight of ash and clinker in fuels varies, as a rule, from 3 to 20 per cent., and yet how seldom is the amount determined or thought of. It should certainly be measured now and then, and regularly if possible, as a check. Shrewd business men, if they know it, are not in the habit of paying 1s. to 2s. per ton for useless dirt.

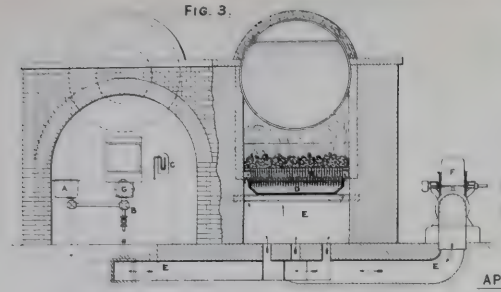
The trials made by Mr. Michael Longridge, of Manchester, are especially valuable, and his interesting annual reports are year by year more complete. Ten years ago nothing so comprehensive was undertaken. He is doing good work in calling attention to the great waste of money in heat and fuel continually going on. The Association Alsacienne des Propriétaires d'Appareils à Vapeur at Mulhouse have also published boiler tests annually, in the same systematic manner, for the past thirty years. To make and work out experiments properly requires a great deal of time, and is

To illustrate Mr Henry White's Paper on "The Coal Electric Miners' Lamp."



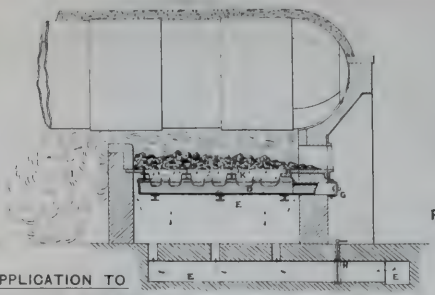
To illustrate Mr Bryan Donkin jun^rs Paper on "Steam Boilers with forced Blast &c."

FIG. 3.



FRONT ELEVATION CROSS SECTION

FIG. 4.



LONGITUDINAL SECTION

APPLICATION TO EGG-ENDED BOILERS

REFERENCES.

A—Water Cistern, with Ball Cock to supply Trough.
B—Drain.
C—Air pressure Gauge.
D—Water Trough.
E—Air under pressure from ½ in. to 1 in
F Fan
G Door for Cleaning Trough
H Air Throttle Valve.
I—Gases under Vacuum produced by Chimney.
J—Flue Door.
K Fire Bars.

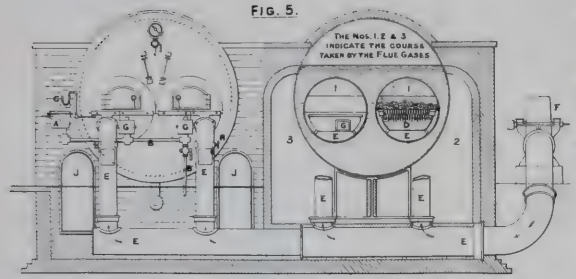
FIG. 1.

CROSS SECTION

FIG. 2.

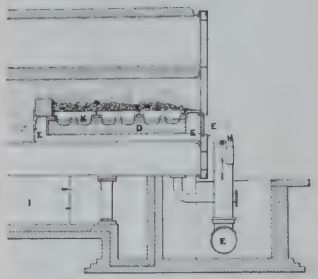
ELEVATION

FIG. 5.



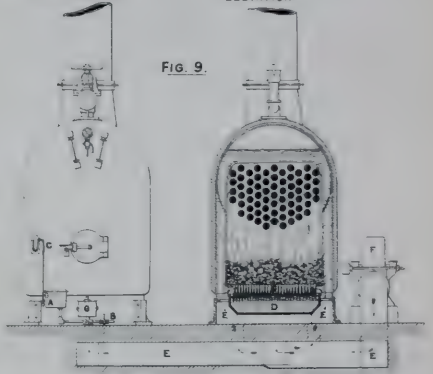
FRONT ELEVATION CROSS SECTION

FIG. 6.



LONGITUDINAL SECTION

FIG. 9.

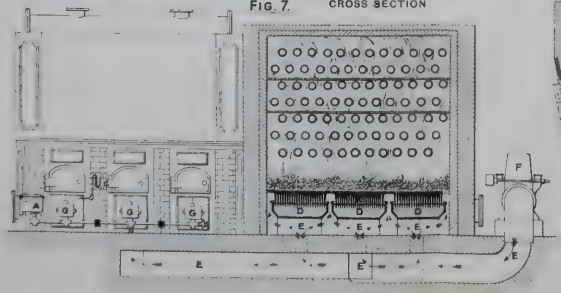


APPLICATION TO LANCASHIRE BOILERS

FRONT ELEVATION

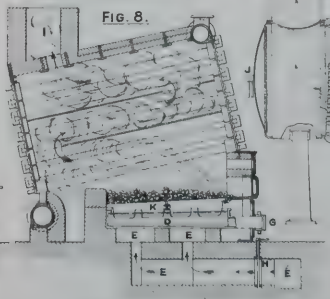
FIG. 7.

CROSS SECTION



LONGITUDINAL SECTION

FIG. 8.



APPLICATION TO TUBULOUS BOILERS

FRONT ELEVATION CROSS SECTION

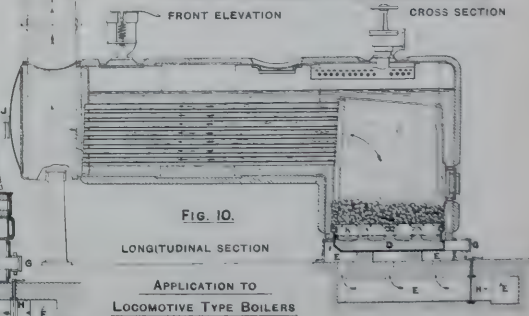


FIG. 10.

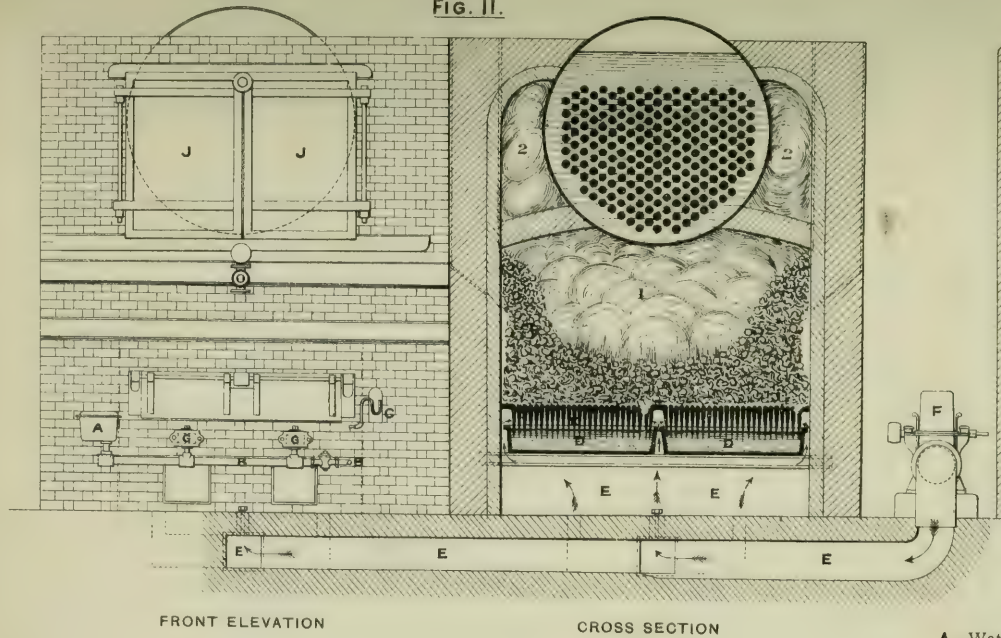
LONGITUDINAL SECTION

APPLICATION TO LOCOMOTIVE TYPE BOILERS FOR FACTORIES

Federated Institution of Mining Engineers
Transactions 1892-93

To illustrate Mr. Bryan Donkin jun.rs Paper on "Steam Boilers with forced Blast, &c."

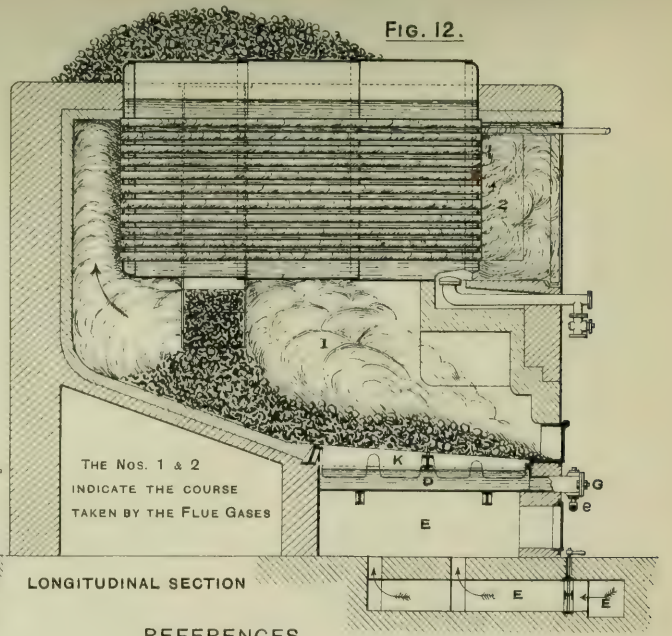
FIG. 11.



FRONT ELEVATION

CROSS SECTION

FIG. 12.



LONGITUDINAL SECTION

APPLICATION FOR BURNING REFUSE IN TUBULAR BOILER

REFERENCES.

A—Water Cistern, with Ball Cock to supply Trough.
B—Drain.
C—Air-pressure Gauge.
D—Water Trough.
E—Air under pressure from ½ in. to 1 in
F—Fan.
G—Door for Cleaning Trough.
H—Air Throttle Valve.
I—Gases under Vacuum produced by Chimney.
J—Flue Door.
K—Fire Bars.

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Transactions 1892-93

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fatiguing, and continuous observations made every 10 or 15 minutes for 12 hours are no light task even to young men. Those who have superintended them know the careful preparation needed to draw out the different working sheets for the detailed observations, giving orders to each of the numerous assistants, checking the instruments, etc. Then there is the actual trial, lasting say from 6 a.m. to 6 p.m. without leaving the hot and dirty boiler-house, etc. After this, the other half of the work has to be done, *i.e.*, the results must be all worked out, checked, tabulated, and the report made.

Experimental Boiler Results with Perret Furnaces.

Appendix A contains an accurate test made by Prof. Kennedy, F.R.S., and the author, on a Lancashire boiler at the Bermondsey works. It will be seen that besides all the details, a heat balance-sheet has been made out, accounting for nearly the whole of the heat given to the boiler.

The expression, "pure and dry coal," is used in these experiments; it means that the weight of clinker, ash, and moisture in the coal has been deducted.

Appendix B records various evaporative results from twenty-eight different kinds of fuels, from a number of experiments at the Bermondsey works on a Lancashire boiler. It will be noticed that the weight of water evaporated per lb. of fuel varies from 2 lbs. to 10 lbs., and the percentage of incombustible matter from $2\frac{1}{2}$ to 36 per cent. This shows the importance of considering the latter factor carefully in all boiler tests.

Appendix C is a summary of results with dust coal from the Moss Hall colliery, made in August, 1892.

The Burning of House-refuse.

As the question of burning ash-bin and other refuse is being much discussed at the present time, it may be well to add a few words on the suitability of Perret furnaces to burn such refuse. In London and many other towns refuse is simply burnt to destroy it on sanitary grounds, but without utilizing at the same time the heat of combustion. It is often desirable to burn refuse, and utilize the heat under steam boilers for the production of steam, etc.

Municipal and town authorities often require power for various purposes which in many cases could be utilized for the production of electric light by means of steam engines

Several of the Perret furnaces are placed under steam boilers burning house refuse. Some are at work in the borough of Warrington, under the direction of Mr. Longdin, the engineer, where the refuse is completely consumed without being screened. The arrangement (Figs. 11 and 12, Plate XII.) is worked in connexion with the Manlove and Alliott destructor, and seems to answer perfectly. Of course, the object in burning refuse should be not only to destroy it, but to utilize the hot gases. At Warrington, the refuse is burnt without producing a nuisance, and the clinker formed is about one-sixth of the total weight. The grates are considerably inclined to assist in getting the refuse into, and the clinker out of, the furnaces The large quantity of clinker made is utilized in various ways, foundations for paving, etc. The steam is used in connexion with a system of concentrating sewerage, and for driving steam engines.

At Shrewsbury gas works a Perret furnace burns, from time to time, house-refuse. In many cases house-refuse could be burnt with an addition of 10 or 15 per cent. of dust, coal, or coke, and smaller boilers would be sufficient.

Mr. Watson, in a recent interesting paper read before the British Association, mentions several classes of house-refuse destructors. He speaks of ash-bin refuse being burnt under a boiler at Oldham, where he measured accurately the quantity of water evaporated into steam, etc., and found the temperature of the furnace to vary between 1,650 degs. Fahr. and 2,350 degs. Fahr.

APPENDIX A.

Experiments with the Perret Grate, burning Welsh Coal-dust, with Forced Draught, at the Engineering Works of Messrs. Bryan Donkin & Co., Limited, Bermondsey, S.E.

The boiler was of the Lancashire type, 7 feet diameter, 21 feet long, having two internal tubes, 2 feet 9 inches diameter, with three conical water-tubes in each. The fire-grates were each 1 foot 10½ inches wide, and 4 feet long: together equal to 15 square feet. The forced blast was supplied by a small fan.

The fuel used was the dust from the Nixon Navigation (Welsh) coal of the following analysis:—Carbon, 88·58 per cent.; hydrogen, 4·10 per cent.; sulphur, 0·66 per cent.; nitrogen, 1·00 per cent.; oxygen, 2·76 per cent.; and ash, 2·90 per cent. The heating power of the pure and dry coal as determined by a calorimeter was 15,955 thermal units, equal to an evaporation of 16·5 lbs. of water from and at 212 degs. Fahr. per lb. of coal.

The average cost of evaporating 1,000 gallons of water with dust Welsh coal, costing 8s. per ton delivered, is 4s. 5d., including lighting and banking up.

	January 14, 1887.	July 20, 1888
Steam—		
Pressure of steam by a continuous self-recording gauge	50 lbs. per sq. in.	57 lbs. per sq. in.
Gases—		
Temperature of issuing gases by mercurial thermometer	399 degs. Fahr.	630 degs. Fahr.
Amount of draught in chimney by water-gauge	0·35 in. of water	0·40 in. of water
Air—		
Pressure of air below firebars	0·10 in. of water	0·20 in. of water
Quantity of air theoretically required for perfect combustion of 1 lb. of coal	11·7 lbs.	11·7 lbs.
Quantity of air used per unit of coal burnt, or lbs. of air per lb. of coal	25·0 lbs.	16·0 lbs.
Lbs. of air used to lbs. theoretically required	2·2 to 1	1·37 to 1
Temperature of air entering grate	65·5 degs. Fahr.	69 degs. Fahr.
Temperature of air outside boiler-house	34·8 degs. Fahr.	64 degs. Fahr.
Temperature of air inside boiler-house	68·5 degs. Fahr.	76 degs. Fahr.
Hours—		
Duration of trial: continuous	9 hours.	12 hours.
Fuel—		
Total fuel used, including ashes, clinkers, and moisture	1,387 lbs.	3,332 lbs.
Incombustibles of total fuel (ashes, clinkers, &c.)	7 per cent.	3·8 per cent.
Fuel burnt per hour	154 lbs.	278 lbs.
Fuel burnt per square foot of fire-grate per hour	10·2 lbs.	18·5 lbs.
Fuel burnt per square foot of heating surface per hour	0·29 lb.	0·48 lb.
Moisture in coal on day of experiment	5·5 per cent.	4·2 per cent.
Gases— Analysis of gases leaving boiler taken continuously, percentages by weight — CO_2	12·46 per cent.	17·35 per cent.
CO	0·00 ,,	0·23 ,,
O	10·89 ,,	7·16 ,,
N	73·61 ,,	70·15 ,,
H_2O	3·04 ,,	5·11 ,,
Water—		
Initial temperature of feed-water	48·6 degs. Fahr.	71 degs. Fahr.
Total quantity of water evaporated during experiments; measured by tank	11,497 lbs.	28,427 lbs.
Water evaporated per hour	1,277 ,,	2,366 ,,
Water evaporated per hour per square foot of heated surface from temperature of feed-water	2·4 ,,	4·06 ,,
Water evaporated per lb. of fuel from initial temperature of feed-water	8·3 ,,	8·52 ,,
Water evaporated per lb. of fuel from and at 212 degs. Fahr.	9·9 ,,	10·01 ,,

BALANCE SHEETS OF THERMAL UNITS, PER LB. OF PURE AND DRY WELSH COAL-DUST.

Experiment of January 14, 1887.

HEAT EVOLVED.	Thermal Units.	Per-centages.	HEAT ABSORBED, ETC.	Thermal Units.
Heating power of 1 lb. of coal, pure and dry, as determined by a calorimeter	15,955	68·7	In feed-water evaporated	10,959
		16·4	In issuing gases... ...	2,623
		6·7	Radiation	778
		0·7	Heat in ashes, clinkers, and fuel drawn from grate...	117
		7·5	Unaccounted for ...	1,478
Total	15,955	100·0	Total	15,955

Experiment of July 20, 1888.

HEAT EVOLVED.	Thermal Units.	Per-centages.	HEAT ABSORBED, ETC.	Thermal Units.
Heating power of 1 lb. of coal, pure and dry as determined by a calorimeter	15,955	65·9	In feed-water evaporated	10,530
		23·3	In issuing gases... ...	3,710
		5·1	Radiation	812
		1·2	Lost by incomplete combustion	186
		4·5	Unaccounted for ...	717
Total	15,955	100·0	Total	15,955

It will be seen that these two experiments differ considerably in the rate of fuel burnt per square foot of grate and rate of water evaporated per square foot of heating surface, the same Welsh coal-dust being used in both.

APPENDIX B.

TWENTY-EIGHT EVAPORATIVE RESULTS OBTAINED BY THE PERRET FURNACES WITH VARIOUS FUELS, WITH FEED-WATER AT 80 DEGS. FAHR.

DESCRIPTION OF FUEL.	Lbs. of Water Evaporated per lb. of Fuel.	Percentage of Clinker and Ash.
Best Welsh coal (small), Nixon Navigation ...	9 to 10 ...	2·4
Cheshire coal, Orrel Co.	8·7 ...	2·4
Ordinary Welsh dust	7·9 ...	3·9
„ „ Murrel's	7·8 ...	7·5
Welsh house slack	6·76 ...	7·5
„ smudge	6·5 ...	23·7
Newcastle large coal	7·6 ...	4·5
„ Wallsend dust	7·26 ...	5·3
Chesterton Colliery, small	6·97 ...	10·66
Nottingham, small	6·5 ...	12·9
Vron colliery, small	6·4 ...	9·4
Babbington colliery, smudge	6·3 ...	8·6
Leicester, small	5·7 ...	8·6

Description of Fuel.	Lbs. of Water Evaporated per lb. of Fuel.	Percentage of Clinker and Ash.
Welsh, anthracite peas	7·28	5·3
Clay Cross, small	6·5	10·0
Welsh, anthracite dust	6·35	6·2
Irish, anthracite culm	4·37	19·5
Ordinary coke	6·85	11·9
Best coke breeze	6·5	10·0
Ordinary breeze (London)	5·4	19·6
Dust breeze	4·4	32·5
Pan breeze from under retort furnaces	3·32	28·6
Refuse breeze from ironworks	4·5	17·5
Cinders from Tunbridge	2·4	36·0
Pyrites coal-waste from South Wales	3·68	35·5
Cocoanut shells	3·3	—
House-refuse, (not sorted), from Warrington ..	2·13	23·5
„ „ Chelsea	2·5	35·3

APPENDIX C.

Summary of Experiments made at the Engineering Works of Messrs Bryan Donkin & Co., Limited, Bermondsey, S.E.

The experiments were made on a Lancashire boiler fitted with Perret furnaces the heating surface was 582 square feet; the grate-surface was 24·72 square feet; the average pressure of air under the firebars was $\frac{1}{2}$ inch; the fires were from 5 inches to 9 inches thick; the steam pressure varied from 55 to 59 lbs. per square inch; the mean temperature of the feed-water was 67 degs. Fahr.; the feed-water was measured in a graduated tank; and the fires were clinkered twice during each experiment.

No. of Experiment.	Description of Fuel used, from Moss Hall Colliery.	Duration of Experiment	Appearance of Smoke.	Moisture in Fuel.	Clinker and Ash in Fuel.	Total Fuel used.	Total Water Evaporated.	Gross Evaporation of Water per lb. of Fuel from Temperature of 67 degs. Fahr. of Feed-water.	Gross Evaporation of Water per lb. of Fuel from Temperature of 212 degs. Fahr.
		Hours.		Per Cent.	Per Cent.	Lbs.	Lbs.	Lbs.	Lbs.
1	Dust coal alone	$9\frac{1}{2}$	Very smoky	$6\frac{1}{4}$	12·50	3,136	18,200	5·80	6·80
2	Do.	$9\frac{1}{2}$	Do.	$6\frac{1}{4}$	11·32	—	—	5·86	6·90
3	$\frac{1}{3}$ dust coal and $\frac{2}{3}$ unscreened gas coke breeze	$9\frac{1}{2}$	—	—	16·66	—	—	5·30	6·25
4	$\frac{1}{2}$ dust coal and $\frac{1}{2}$ unscreened gas coke breeze	$9\frac{1}{2}$	—	—	15·17	—	—	5·10	6·00

APPENDIX D.

Steam Boilers.—Daily Return of Fuel and Water, Etc.

No..... Boilers at.....for week ending.....

Date.		Boiler.				Fuel.				Feed-water.				
18 Month.	Date.	Number of Hours under Steam.			Average Pressure of Steam.	Weight of Fuel Burnt. B.	Weight of Clinker and Ashes. A.	Percentage A. of B.		Reading of Water Meter or Tank at o'clock.	Quantity of Water Consumed.	Average Temperature of Water.	No. of Hours Engine Run.	
		From	To	Total Hours	Lbs.	Lbs.	Lbs.	Per Cent.			Lbs.	Degs.	Hours	
Monday ...														
Tuesday ...														
Wednesday ...														
Thursday ...														
Friday ...														
Saturday ...														
Totals ...														

Remarks.—Give date whenever boiler is cleaned or inspected, also when flues are cleaned.

Kind of fuel.....
Temperature of gases, if taken .....
Number of times boiler blown off .....
Damper generally.....open
Lbs. of water evaporated per lb. of fuel (including lighting and banking up) .....

Name of stoker this week ....
Average cost of fuel per ton, delivered into boiler-house .....
Cost of fuel per week ....
Cost of fuel per 1,000 gallons evaporated .....
Inches by gauge glass.....ins. = ...lbs.

Signed and checked by.....

NOTES ON THE OCCURRENCE OF MANGANESE ORE NEAR THE ARENIGS, MERIONETHSHIRE.

By EDWARD HALSE, A.R.S.M.

Mr. Edward Halse communicated the following addendum to his Notes*:—

The following is a complete analysis, by Messrs. Johnson, Matthey, & Co., of a specimen of manganese ore from the principal vein in the district (Plate LXI., Fig. 4):—

	No. 4. Per Cent.	
Peroxide of manganese	55·3	= 48 metallic manganese.
Protoxide of manganese	12·3	
Oxide of iron	7·2	
Oxide of cobalt	1·4	
Alumina	6·8	
Magnesia	0·4	
Lime and barytes	traces	
Sulphur	0·1	
Phosphoric acid	0·1	= ·043 phosphorus.
Silica and insoluble matter ...	10·0	
Combined water	5·8	
Loss	0·6	
	100·0	

The following are partial analyses of ore from some of the other veins :—

	No. 5. Per Cent.	No. 6. Per Cent.	No. 7. Per Cent.
Metallic manganese	49·13	48·00	38·40
Silica	2·14	9·00	7·92
Phosphorus	0·16	0·38	0·26

The metallic manganese varies from 38 to 54 per cent. (see page 946), the silica and insoluble matter from 2 to 10 per cent., and the phosphorus from ·043 to 1·608 per cent.

* *Trans. Fed. Inst.*, vol. iii., page 940.

Since the notes were written, a paper* by Mr. Charles Vattier was read before the French Society of Civil Engineers. This writer asserts that manganese ores occur in Chili in veins, traversing Jurassic limestone and other rocks, varying in thickness from $3\frac{1}{4}$ to nearly 10 feet, as well as in seams and irregular masses. The author observes that generally, at a very slight depth, the quality and contents of the ore are lowered appreciably, the gangue becomes more quartzose, red hydrated oxide of iron taking the place of manganese oxide, and in many cases, notwithstanding superficial indications of great richness, the ore diminishes considerably, and even disappears altogether at a depth of $45\frac{1}{2}$ to $48\frac{3}{4}$ feet.

* "Le Chili; Mineur, Metallurgique, Industriel."—*Mémoires de la Société des Ingénieur Civils*, series 5, vol. xlv., pages 37-140.

NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE,

NOVEMBER 5TH, 1892.

MR. J. B. SIMPSON, PRESIDENT, IN THE CHAIR.

The minutes of the last General Meeting were read and confirmed.

The SECRETARY reported the proceedings of the Council, and said that the question of lighting the Wood Memorial Hall and lecture-room by electricity had been referred to a committee.

The following gentlemen were elected, having been previously nominated :—

HONORARY MEMBER—

Prof. HENRY STROUD, D.Sc., Durham College of Science, Newcastle-upon-Tyne.

MEMBERS—

Mr. ALEXANDER MATHIESON, Colliery Manager, Hetton Colliery, Carrington, near Newcastle, New South Wales.

Mr. CLAYTON E. J. MITCHELL, Civil Engineer, 27, Mosley Street, Newcastle-upon-Tyne.

Mr. JAMES NEWTON, Colliery Manager, Whitehaven Colliery, Whitehaven.

Mr. GEORGE OLDHAM, Engineer, 25, Western Hill, Durham.

Mr. T. A. TATE, Colliery Manager, Bunker Hill, Fence Houses.

ASSOCIATE MEMBERS—

Mr. GEORGE BLAXLAND ATKINSON, Timber Merchant, Maritime Buildings, Quay, Newcastle-upon-Tyne.

Mr. J. P. CORNETT, Managing Director, Ford Paper Works, Hylton, Sunderland.

Mr. ARTHUR PATRICK DODDS, Iron Merchant, 13, Dean Street, Newcastle-upon-Tyne.

Mr. CLAUDE BOWES PALMER, Assistant Colliery Manager, Usworth Colliery, Washington, R.S.O.

ASSOCIATES—

Mr. THOMAS BERRY, Overman, Bagnall's Houses, Swalwell, R.S.O.

Mr. ROBERT CARR, Under Manager, Usworth Colliery, Washington, R.S.O.

Mr. WILLIAM SEVERS, Under Manager, Beamish View, *via* Chester-le-Street.

STUDENTS—

Mr. WILLIAM HENRY MASCALL, Mining Student, 27, South Church Lane Bishop Auckland.

Mr. GEORGE WALL, Mining Student, South Church, Bishop Auckland.

The following gentlemen were nominated for election:—

MEMBERS—

Mr. A. D. DIXON-BROWN, Electrical Engineer, 27, Mosley Street, Newcastle-upon-Tyne.

Mr. P. S. REID, Mining Engineer, 20, John Street, Adelphi, London, W.C.

Mr. CEDRIC VAUGHAN, Managing Director, Hodbarrow Iron Ore Mines, Millom, Cumberland.

ASSOCIATE MEMBER—

Mr. ALFRED ALLHUSEN, Managing Director, Newcastle Chemical Works, Musgrave House, Gateshead-upon-Tyne.

ASSOCIATES—

Mr. JOHN BUGLASS, Colliery Manager, Stobswood Colliery, Acklington, Northumberland.

Mr. FRANK HOWES, Under Manager, Singareni Colliery, Hyderabad, Deccan, India.

Mr. CHRISTOPHER HUNTER, Engineer, Broomhill Colliery, Acklington, Northumberland.

Mr. SIMON PEDELTY, Engineer, Clara Vale, Ryton-upon-Tyne.

Mr. JOHN GLADSTONE RAMSAY, Deputy, Page Bank Colliery, Willington, Co. Durham.

DISCUSSION UPON MR. W. COCHRANE'S PAPER "ON A DUPLEX ARRANGEMENT OF CENTRIFUGAL VENTILATING MACHINES."*

Mr. COCHRANE said he had nothing to add to his paper, but he thought it was understood, when the discussion on it was adjourned, that Mr Wallau would submit a communication on the subject.

Mr. THOS. BELL asked if there was a Capell fan working at any neighbouring colliery with the exception of that at East Howle?

Mr. M. WALTON BROWN thought not.

Mr. COCHRANE said that there were some large Capell fans working in Germany.

Mr. M. WALTON BROWN said that the report of the Prussian Firedamp Commission showed that ventilating fans in Germany rarely gave more than 18 per cent. of useful effect. Prof. Bedson was translating this report for the Institute, and it would shortly be published.

* *Trans. Fed. Inst.*, vol. ii., page 483.

Mr. COCHRANE said high results were claimed for the Capell fan. Of course a good deal depended on the conditions of the experiments; he understood that fan was dealing with volumes of 100,000 to 250,000 cubic feet of air per minute under 6 inches of water-gauge.

Mr. A. L. STEAVENSON said the inventors of fans were apt to deceive themselves, and each one got more wonderful results than another. He recently went to Sheffield to assist in taking the measurements of the air and water-gauge in testing a fan, but he was certain that the measurements of air were too large, and consequently the results could not be accepted as correct.

DISCUSSION UPON MR. R. S. WILLIAMSON'S PAPER ON "THE CHANDLER FAN," AND MR. M. WALTON BROWN'S PAPER ON "THE RATEAU VENTILATOR."*

Mr. M. WALTON BROWN said the results of the experiments recorded in these papers on the Chandler and Rateau ventilators emphasized the importance of the trials being made with every precaution to obtain accuracy under similar conditions. The efficiency of the Chandler fans varied from ·57 to ·70, and there were no means of accounting for this material difference. The experiments on the Rateau fan were made by Mr. de Castelnau. One did not like to say they were altogether wrong, but as the mechanical efficiency varied from ·46 to ·72, it proved that the mode adopted of measuring the air at the top of the expanding chimney was not satisfactory.

Mr. COCHRANE enquired whether Mr. Brown thought that the Rateau ventilator was as efficient a machine as the other ventilators with which he was acquainted ?

Mr. M. WALTON BROWN replied that he brought the experiments upon the Rateau fan before the members of the Institute as being made on a fan for which a high efficiency was claimed, but he did not think it was more efficient than others of similar dimensions.

Mr. R. S. WILLIAMSON wrote in reply to Mr. Walton Brown's remarks upon the different efficiencies of the two Chandler fans that he experimented upon, that both fans were of the open-running type. The horizontal engine of 13 inches diameter and 26 inches stroke was suitable for 30 indicated horse-power, and, with this load, would absorb at least 10 per cent. of the power in friction; including the belt the 10 feet fan absorbed 21·13 indicated horse-power, and the 6½ feet fan 15·17 indicated horse-power, consequently, therefore, the engine and belt friction at this

* *Trans. Fed. Inst.*, vol. iii., page 171 and page 410.

light load was about 20 per cent. The mean effective pressure in the cylinder when the engine was indicating 21·13 horse-power was only 11·5 lbs. per square inch, and it is clear from this fact, that the engine could not run with less than 15 per cent. of frictional resistance when indicating 21·13 horse-power, therefore, when indicating 15·17 horse-power the frictional resistance could not be less than 20 per cent.; and both fans would give much higher efficiencies, with smaller engines, working at their full power. It is an established fact that engines work with the same frictional resistance with either heavy or light loads, and that the friction does not increase with the load. The friction of engines is practically constant at all reasonable loads, which accounts for part of the loss of efficiency in the small fan, say to the extent of 5 per cent. Further, the small fan had not such well-formed blades as the 10 feet fan. The space of brickwork round the blades was not so large, and the air did not pass away so freely, which would account for a further decrease of efficiency in the small fan. The fans still continued to afford entire satisfaction, and had not cost a single penny for repairs since starting. The inventor had, since the reading of the writer's paper, made several larger fans, which have given great satisfaction.

DISCUSSION ON MR. M. WALTON BROWN'S PAPER ON "THE KÖNIG DIFFERENTIAL WATER-GAUGE."*

Mr. M. Walton Brown exhibited a König water-gauge coupled with an ordinary type of water-gauge. He had been able to try the König water-gauge on a fan drift in comparison with an ordinary water-gauge. The great advantage of the König water-gauge, in addition to its giving an exaggerated scale of say 9 to 1, was the fact the readings were more easily taken, owing to the comparative steadiness of the fluid, as compared with the fluctuations in the levels of the water in an ordinary gauge, and this allowed of much more than nine-fold accuracy in the readings.

DISCUSSION UPON MR. W. N. ATKINSON'S PAPER ON "THE REPORT OF THE AUSTRIAN FIRE-DAMP COMMISSION."†

The Secretary read several portions of the paper to which the author desired to draw especial attention.

Mr. Thos. Bell (H.M. Inspector of Mines, Durham) said that the quantity of air per person recommended by the Austrian Commission was

* *Trans. Fed. Inst.*, vol. iii., page 452. † *Ibid.*, vol. iii., page 531.

far below the quantities employed in this country. In Durham—in the coal-mines of his district—the annual returns for the past year showed that there was on average 347 cubic feet of air per minute circulating in these mines per person employed underground; and 420 cubic feet per minute in the Cleveland ironstone mines. These figures did not, however, take into account the number of horses and ponies employed, but allowing for these, the ventilation of the mines in his district was very far in excess of the recommendations of the Austrian Commission.

The PRESIDENT said the figures given in the paper (70 cubic feet where little fire-damp was produced, 105 cubic feet for the second class where there was a moderate amount, and 140 cubic feet in the third class) were only recommendations; the actual state of things was possibly worse.

Mr. M. WALTON BROWN thought the recommendations were made in order to provide against the use of an excessive volume of air on account of the alleged dangers which might arise from the action of larger volumes of air in drying coal-dust.

Mr. SIMON TATE said there was a reference in the paper to the fact that 20 per cent. more air was found in the returns than in the intakes. Was that due to the expansion of the air in the mine or to the addition of gas?

The PRESIDENT thought it would be unnecessary to bring the paper before another meeting. It was a record of the work of the Austrian Commission, and it would not be necessary to discuss it further.

DISCUSSION UPON MR. W. HY. CHAMBERS' PAPER ON "THE PUMPING APPLIANCES USED IN THE SINKING OPERATIONS AT THE CADEBY NEW WINNING."*

Mr. A. L. STEAVENSON thought that the simpler they could have everything for use in sinking pits the better it would be. His impression was that they could have nothing simpler under usual circumstances than the old system of horizontal engine with quadrants.

The PRESIDENT agreed with Mr. Steavenson. He had had the discomfort recently of dealing with 2,000 gallons of water per minute at a depth of 50 fathoms; and he had been able to overcome this feeder with a 24 inches set of pumps worked by bell cranks. There would scarcely have been room in the shaft to have dealt with this quantity of water by Mr. Chambers' system. The paper would have been more interesting and useful if it had given some account of the cost of the system described; it was questionable whether, all things considered, the old system would not have been cheaper and more efficient.

* *Trans. Fed. Inst.*, vol. iii., page 513.

Mr. W. C. Blackett said no doubt portable steam-pumps had been successful in certain cases, where all the pipes and even the pump itself came in for other purposes afterwards.

Mr. T. E. Forster said he thought Mr. Chambers' principal reason for adopting the system described was the extreme badness of the ground; he dared not place a heavy engine on the top.

Mr. W. Cochran Carr wrote that he had been using a Denaby sinking pump with a steam cylinder of 22 inches diameter and 36 inches stroke, with a double-acting ram of 13 inches diameter. He regretted that this pump had not been working long enough to give a definite opinion about it, but so far it had worked satisfactorily, and was the best direct-acting pump of its sort he had seen. The following were its strong points in his opinion:—(1) It occupied little room in the shaft, and had the clacks, steam-chest, pipes, etc., so placed that when suspended it hung quite plumb; (2) it had large valve area, which enabled it to make a greater number of strokes per minute with less vibration than other pumps of the same stroke; (3) in case of necessity it would work with the cylinder submerged to a depth of several feet, even when exhausting direct into the water; (4) the general arrangements of the steam-chest and valves were satisfactory, there being no position which the piston could occupy in which it would not start, even if under such circumstances as described in the third remark; (5) the first joint next to the pump for water and steam was expansive, and allowed for any movement and prevented the breakage of pipes, which he thought had always been a difficulty with other pumps of similar kinds; and lastly the Denaby pump did not take as much steam as an Evans pump which was used in the same shaft, although the former was an inch larger in diameter of cylinder, the stroke being the same in each. He regretted that he had not had more experience with it, so as to give details as to the quantity of water actually pumped, and to compare its work with the Evans pump working in the same shaft.

Mr. W. Hy. Chambers, in reply, wrote that he concurred with Mr. Steavenson's opinion that the more simple everything could be made for use in sinking pits the better it would be. Having had extensive experience of direct-acting pumps and also of the old system of pumping with horizontal engines, working quadrants, the comparison in his opinion was very largely in favour of the more recent system, which was accompanied with marked simplicity. He did not stand alone in this opinion, as, since its introduction at Cadeby, the method had been adopted very extensively in preference to the old-fashioned system, and there are now, to the writer's knowledge, no less than nine different

kinds of pumps on this principle, or as near as can be without infringing on the patent rights of Messrs. Bailey & Co. The first ones of the same style were made by Messrs. Evans, of Wolverhampton, for the Griff Colliery, after Mr. E. Bainbridge and Mr. Melly endeavoured to make terms with the Denaby Company for hiring the pumps which were then done with. Mr. Melly wrote an article some time ago to the *Colliery Guardian* expressing his perfect satisfaction with the principle adopted; and with one or two exceptions this had generally been the case wherever the system was tried. With regard to Mr. Simpson's remarks, stating that he had no difficulty in dealing with 2,000 gallons of water per minute at a depth of 50 fathoms with a 24 inches set of pumps, and that there would scarcely have been room in the shaft to have dealt with this water by the newer system; the fact was that four times that amount of water was elevated from a depth of 60 fathoms by that system, and any one would have understood the position if four 24 inches sets of pumps had to be worked in a 16 feet sinking shaft, and the impossibility of putting in tubbing while they were working. With regard to the cost of the system described, the whole of the machinery, including pumps and pipes, was sold at a discount of not more than 10 per cent. This loss and the cost of replacing breakages was under £1,000. The labour employed in working the pumps was a fitter at 6s., a boy at 2s. 6d., and the man at the capstan-engine at 4s. per shift. The banksmen assisted the fitters to build up pipes on the top as required, when the pumps were lowered, and this entailed no extra cost. Mr. T. E. Forster's surmise was quite correct, that the principal reason for adopting the system described was the unfitness of the ground to carry heavy weights, necessitating the striking out a new line to cope with the difficulty. The writer could only say, after experience of both methods, that if any similar work be entrusted to his direction, and were the ground to be as bad as in this instance, or good enough to enable heavy lifts to be used, he would never of his own free will resort to such a costly, clumsy, and dangerous system as the placing of such tremendous weights of useless metal, as necessitated by the old-fashioned, and soon to be obsolete, system of lifting water in sinking pits by means of spear-rods and buckets, with its heavy train of strong gearing, engine, and powerful capstans, which were almost invariably consigned to the scrap heap, and sold as old iron on the completion of the work.

There being no further remarks offered, the meeting terminated.

NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,
HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE,
DECEMBER 10TH, 1892.

MR. J. B. SIMPSON, PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the previous General Meeting, and reported the proceedings of the Council. The agreement with the North-Eastern Railway Company as to the use of the passage adjoining the lecture-theatre had been sealed. Arrangements were being made for the electric-lighting of the Wood Memorial Hall and the lecture-theatre, etc.

The following gentlemen were elected, having been previously nominated:—

MEMBERS—

Mr. A. D. DIXON-BROWN, Electrical Engineer, 27, Mosley Street, Newcastle-upon-Tyne.

Mr. P. S. REID, Mining Engineer, 20, John Street, Adelphi, London, W.C.

Mr. CEDRIC VAUGHAN, Managing Director, Hodbarrow Iron Ore Mines, Millom, Cumberland.

ASSOCIATE MEMBER—

Mr. ALFRED ALLHUSEN, Managing Director, Newcastle Chemical Works, Musgrave House, Gateshead-upon-Tyne.

ASSOCIATES—

Mr. JOHN BUGLASS, Colliery Manager, Stobswood Colliery, Acklington, Northumberland.

Mr. FRANK HOWES, Under Manager, Singareni Colliery, Hyderabad, Deccan, India.

Mr. CHRISTOPHER HUNTER, Engineer, Broomhill Colliery, Acklington, Northumberland.

Mr. SIMON PEDELTY, Engineer, Clara Vale, Ryton-upon-Tyne.

Mr. JOHN GLADSTONE RAMSAY, Deputy, Page Bank Colliery, Willington, Co. Durham.

The following gentlemen were nominated for election :—

MEMBERS—

Mr. THOMAS ALEXANDER ALLAN, Consulting Mining Engineer, Brightlands, Richmond Hill, Surrey.

Mr. THOMAS BREAKELL, Mining Engineer, 92, Denmark Road, Greenheys, Manchester.

Mr. WALLACE THORNEYCROFT, Mining Engineer, 6, Dixon Street, Glasgow.

Mr. HENRY THOMAS WALES, Mining Engineer, Pontypridd, South Wales.

ASSOCIATE MEMBERS—

Mr. JAMES ERNEST BLENKIRON, Mining Engineer, West House, Arkengarthdale, Richmond, Yorkshire.

Mr. J. CARTMEL RIDLEY, Steel Manufacturer, 3, Summerhill Grove, Newcastle-upon-Tyne.

ASSOCIATE—

Mr. JOHN SPRY, Surveyor, Murton Colliery, *via* Sunderland.

CONFERENCE OF DELEGATES OF CORRESPONDING SOCIETIES OF THE BRITISH ASSOCIATION FOR THE ADVANCEMENT OF SCIENCE, EDINBURGH, 1892.

The report of the proceedings of the Corresponding Societies Committee of the British Association was read, and the report of Prof. J. H. Merivale, the delegate of this Institute, as follows :—

NEWCASTLE-UPON-TYNE,
AUGUST 22ND, 1892.

TO THE PRESIDENT AND COUNCIL OF THE NORTH OF ENGLAND INSTITUTE.

DEAR SIRS,

I attended the two meetings of the Delegates of Corresponding Societies, held at Edinburgh on the 4th and 9th inst. There was a good muster, but there was very little on the agenda-paper of special interest to the North of England Institute of Mining and Mechanical Engineers. I had hoped to have been able to have brought before the meeting some report of the experiments of the Institute on Flameless Explosives, but the Durham strike has unfortunately prevented this.

The Earth-tremors' Committee have examined a large number of recording instruments, and a very simple form, designed by Professor G. Darwin, consisting essentially of a pendulum with double-suspension mirror is recommended provisionally. Professor Darwin has offered to lend and set up one of these instruments, and the Secretary of the Committee (Mr. Davison, an old student of the Durham College of Science) has agreed to look after it and observe its ways and manners at Birmingham during the ensuing year. If it answers the purpose, the Committee

will be in a position to finally recommend it in their report at the next meeting of the British Association at Nottingham. When this is done, a plan for a network of observing stations over the country may be elaborated.

The Underground Temperature Committee reported the observations made in an exceedingly dry borehole in America : where the rise of temperature was found to be 1 deg. Fahr. per 86 feet, a slower rise than usual. The borehole was one made by the American system, which brought up no core; an accurate section therefore could not be obtained, a matter of some importance in this connexion. The boring was to be continued to a depth of 6,000 feet for scientific purposes, and would then, I believe, be the deepest borehole in the world.

Yours faithfully,

JOHN H. MERIVALE.

INTERNATIONAL ENGINEERING CONGRESS, CHICAGO.

A letter was read from the General Committee of the Engineering Congress, inviting the members of the Institute to attend the Congress, which will hold its session from July 31st to August 5th, 1893, inclusive.

The Rev. G. M. CAPELL then read the following "Observations on Fans of Different Types working on the same Upcast Shaft."

OBSERVATIONS ON FANS OF DIFFERENT TYPES WORKING ON THE SAME UPCAST SHAFT.

By the Rev. G. M. CAPELL.

In 1884, when testing various types of fans, the writer was able to make a test of a Capell fan and a Guibal fan working under similar conditions on the same fan drift. The Capell fan was 5 feet in diameter, while the Guibal fan was 16 feet, but the results were not sufficiently clear to arrive at any definite conclusions.

One point was prominent, and this was recorded at the time, viz., that there was a different water-gauge in the Guibal fan when passing the same volume of air as the small Capell fan. Unfortunately the arrangement was removed, rendering further experiments impossible; but the writer made further tests on small fans, and found the same variation in a less marked degree.

There are now many Capell fans working on collieries, replacing large Guibal fans, and other types; and the observations of others on some of the principal of these fans may be now usefully recorded.

The point of the early tests at Homer Hill colliery comes out very clearly, and the lower gauge, for a given work in air, shows itself wherever the Capell fan has replaced a Guibal fan. There is only one case of a Capell fan and of a Schiele fan on the same shaft, and the difference in gauge has been fully noted in Mr. Maurice Deacon's account of his observations on a Capell fan and a Schiele fan working under similar conditions, at Shirland colliery, where the Capell fan showed the same low gauge for a given volume of air.*

The new type of Capell fan differs entirely from anything in use before, consisting of two distinct fans built up into one, so that the air of

* *Trans. Fed. Inst.*, vol. i., page 287.

the first is discharged into the second, its *vis viva* being destroyed by expansion into the larger area, under greater water-gauge, by reason of the greater velocity in the outer fan. The writer maintains that the action of the two machines, in dealing with air, is similar to that of a compound engine dealing with steam. The double arrangement gets more work out of the steam. But work in a fan is, to a great extent, the *vis viva* of the air, and if this is used up there is more effective work got out of the Capell machine; just as the slow, soft discharge from a compound engine, as compared with the rapid noisy discharge of a simple high-pressure engine tells us that more work has been taken out of the steam. Prof. Guibal got the right idea in his évasée chimney, but if the discharge of six revolving évasée's can be obtained, which is the point aimed at in the construction of the Capell fan, a still higher result may be expected than can be had with a fixed expanding chimney.

When the writer is asked how he can account for the differences shown, he answers that he has noted that some fans give more local action in gauge than others, *i.e.*, have a higher gauge near their inlets, but that extra gauge is not effective at a distance from the fan.

The writer remembers a case at a colliery near Birmingham, where a small Schiele fan, under 1 inch of water-gauge, produced 21,000 cubic feet of air. A large Guibal fan was put on, and with 1 inch of water-gauge gave about 35,000 cubic feet of air. On enquiry the writer found that, whereas the small fan gave about 0·50 inch of water-gauge at the bottom of the deep shaft, the large fan gave very little less water-gauge than on the surface. This fact points to the possibility of a variation in volume for gauge where there is a fan of large exhausting capacity, capable of throwing its gauge a long way back into the mine.

The first comparative tests that the writer will give of a Capell fan and Guibal fan will be those at the Prosper I collieries, near Borbeck, in Westphalia. As the guaranteed effect was higher than had ever been before obtained in the history of fans, the writer had special precautions taken in the measurements of the air passed by the Guibal fan, and as will be seen, the writer at once took the guarantee up. The state of things was as follows:—There was a Guibal fan 40 feet $6\frac{3}{4}$ inches in diameter, and 8 feet 4 inches wide; the volume of air passing was 59,340 cubic feet of air per minute, under a water-gauge at fan of 3·15 inches, and the volume of air required was 116,490 cubic feet per minute. The gauge can be easily calculated by the square of the volumes—$59{,}340^2 : 116{,}490^2 :: 3{\cdot}15 : 12{\cdot}13$ inches being the gauge required for 116,490 cubic feet of air with the Guibal fan.

The tests of the Capell fan at the Prosper I colliery, conducted by Mr. G. Herbst, engineer to the Prussian Fire-damp Commission, came out as follows:—

	Ft. Ins.
Diameter of the Capell fan	12 6
Width „ „	6 0
Two inlets, each	7 3
Revolutions of engines	87·5 per minute.
„ of fan	323 „
Water-gauge in drift	10·7 inches.
Cubic feet of air per minute	127,574
Horse-power in air	214·9
„ of engines	384·6
Useful effect	55·6 per cent.

Taking the squares of the volumes from the Guibal fan, it is found that its water-gauge for 127,574 cubic feet would have been 14·5 inches, or 3·8 inches more than the Capell fan, or a difference of 78 horse-power in the air. The writer simply records the figures as given to him.

Another marked case was at the Gneisenau colliery, near Dortmund, also in Germany. The Guibal fan, 40 feet 6¾ inches in diameter, with a drift gauge of 1·60 inches, passed 60,010 cubic feet per minute. The writer was asked to give 105,900 cubic feet per minute at 200 revolutions. For this purpose he designed a double-inlet fan, 11 feet in diameter, 7 feet wide, with two inlets each 7 feet in diameter. The results came out as follows:—

Revolutions of fan per minute	168
Water-gauge in drift	1·70 inches.*
Cubic feet of air per minute	84,720

The results of trials of the two fans, at nearly the same water-gauge being:—

	Guibal Fan.	Capell Fan.
Water-gauge ...	1·60 inches	1·70 inches.
Volume ...	60,010 cubic feet.	84,720 cubic feet.

The gauge difference is obvious.

At Shirland colliery, the Schiele fan is 5 feet 3 inches in diameter, with double inlets; and the Capell fan, 12 feet 6 inches in diameter, with single inlet. Capell fan, 1 inch drift gauge—60,084 cubic feet. Schiele fan, 1 inch drift gauge—43,699 cubic feet. Capell fan, 3·25 inches drift gauge—118,241 cubic feet. Taking the squares of volumes, to get 118,000 feet, the Schiele fan would have required no less than 7·55 inches of water-gauge. These figures are not the results of the writer's own observations, as he has before stated.†

* The volume of air being very large in proportion to the fan's cubic capacity, the manometric effect is low.

† *Trans. Fed. Inst.*, vol. i., page 289.

At Coalpit Heath colliery, near Bristol, there are on the same upcast a Guibal fan, 16 feet in diameter and 5 feet 3 inches wide, and a Capell single-inlet fan, 12 feet in diameter, 5 feet wide, with inlet 7 feet in diameter The results came out thus:—

	Guibal Fan.	Capell Fan. (1)	Capell Fan. (2)
Revolutions per minute	120	150	223
Water-gauge (inches)	2·50	1·75	4·50
Mean air-speed in drift per minute (feet) ...	830	885	1,309
Area of drift (square feet)	50	52	52
Volume per minute (cubic feet)	41,500	46,020	68,068

The drifts were divided into squares with wires, and the anemometers were used for equal periods in each square. The above observations were taken by Mr. G. Hewitt and the late Mr. G. Nasmyth, of Coalpit Heath colliery.

The writer could give many other observations from German collieries, and when the paper is discussed he hopes to do so. All experiments show the same sort of variation in gauges required.

In conclusion, and for comparison, the writer gives the water-gauge of fan and furnace at Silverhill colliery belonging to the Stanton Iron Works Company.*

The furnace water-gauge taken below was 1·40 inches, and the mean volume at returns was 110,000 cubic feet per minute.

The Capell fan, 12 feet 6 inches in diameter, with two inlets 7 feet 6 inches in diameter, gave the following results:—

	(1)	(2)
Revolutions	123	132
Water-gauge (bank)	1·40	1·50
„ measured below, at same place as with furnace (inches)	1·30	1·40
Volume per minute at returns (cubic feet) ...	110,920	129,150

The shafts are each 400 yards deep and 15 feet in diameter. Apart from shaft friction, the fan gets more air than the furnace at the same water-gauge below ground; and an equal quantity at the same water-gauge in the drift, as with the furnace water-gauge below ground.

The writer's object is to give merely the results of his comparisons, and he does not propose to make any further remarks, but hopes to hear what others have to say upon the subject.

* *Trans. Fed. Inst.*, vol. ii., page 549.

Prof. A. S. Herschel (Slough) wrote that it had hitherto seemed to be quite inexplicable to mining engineers (and he had been entirely at a loss till now for any—even weakly defensible—interpretation of the remarkable mode of action observed in double-chambered fans, of which Mr. Capell had produced in his paper from his experience with his fans, so many reliable examples) that weaker drift water-gauges are required with them than with single-chambered fans, or as compared with furnaces, to produce the same volume of air-current in one and the same mine's unchanged conditions of resistance. Although so seemingly impossible to reconcile with the theory that a thin-plate orifice's flow with *vena contracta* is a true equivalent of any such unvarying resistance, and that the relation of air-volume to water-gauge should therefore be as fixed as in the case of such an orifice, with whatever description of fan the water-gauge is produced which makes the current; yet that the contrary experiences from this are not the results of ill-made observations, or illusory determinations, was proved by the multiplied examples of such abnormal features which the comparisons of different fans, working on the same mine with each other, had during the last few years brought prominently forward. The singularity in question had indeed been for some time the theme of controversies among users of ventilating fans, ever since the first applications were made successfully on a large scale to mine ventilation, of the Capell double-chambered fan. At an early period of these experiments, in August 1889, Mr. Capell first compared together a Guibal fan and a Capell fan working on the same mine beside each other, at Coalpit Heath colliery, near Bristol. In that experiment, when a 16 feet Guibal and an 11 feet enclosed single-inlet Capell fan were run upon a mine together at the same speed (80 revolutions in a minute), for which it was found that the two fans working separately produced the water-gauge depressions in their drifts of 1·10 inch and 0·83 inch respectively, the Capell fan drew air from the mine, in spite of its low water-gauge, in a measurable current. At 88 and 90 revolutions of the Capell fan, while the Guibal fan contined to run at 80 revolutions, the conditions of proportion in the air-draughts were reversed, hardly any air-current then passing through the Guibal fan, although the water-gauge of the Capell fan in its drift, when working separately, only just attained 1·00 inch at 90 revolutions. This feature which the Capell fan was then found to show of excelling other fans in lowness of drift water-gauge, for a given air-volume on a given mine resistance, had been so frequently verified and confirmed by subsequent comparisons with other fans tested beside it on the same mine resistances, that the well-recorded

proofs of the peculiarity cited by Mr. Capell in his present paper are only a select few of the most certain evidences which had been very plentifully noted of the fan's very singular and abnormal-looking property. In the first published experiments made with the fan by Mr. D. K. Clark, in 1884, it exhibited, like a Guibal fan, nearly the full double-power water-gauge due to blade-tip velocity, at its air inlet. Calculating the horse power in the air by the ordinary formula, with this water-gauge and volume, it was found to exceed by many per cent. the amount of power used to drive the fan, proving (for this is all that can be concluded positively from the observation) that by a small permanent output of actual from latent energy, a considerable fraction of the air-current which passed through the fan was set flowing idly, or perpetually, as a companion stream to the main current, rather than drawn along the route by mechanical work against mechanical resistance. So far as the air's oblique impact, or dash against the fan's outer wing-blades, in spouting from the port-holes of the inner fan-chamber assists the fan-body's revolution, some partial measure of the air's volume may, in fact, be regarded as coursing through the fan in an endless circulation of closed stream lines, requiring no mechanical work to maintain their self-recuperative motion ; for just such streams, endlessly enduring flux and reflux of actual and potential energy of motion present themselves in fluid vortex-filaments as a kind of perpetual orbitation of fluid-matter rings round a core or region of relatively lower pressure than the encompassing retaining pressures prevailing in the surrounding body of the fluid mass. As the growth of physical knowledge has swiftly advanced with great strides in recent years, and as the need of placing energy and matter, in a rigid prosecution of all the sciences (and especially in the leading column ranks at the van of their advance, of the exact sciences) on an objective level of the study of nature with each other as the universal mind and body of nature's operations, has of late years grown to be strongly felt and very clearly recognized, this current flow of constant freedom and of permanent deliverance from mechanical propulsion in the companion current through the fan, may now easily be perceived to owe its franked liberty of passage through the fan's inlet-tube and chamber-channels to a double, or taking-and-return transaction of work performance and of work-resumption between the current and the driving-power, on the two sets of fan-blades fixed firmly to the driving axle; which compensating pair of work-compounding actions eliminates the power's concernment with the stream-flow, while it therefore also, at the same time, sets free the idle current from friction's drags against the inlet's sides

and the fan-chambers' wings and blades past which it races, which are all confronted singly and collectedly to the driving-power directly. The consumpt of power by friction in passing these obstructions is thus arrested undiscernibly in the same process with the two transaction-dealings with the driving-power which are performed visibly and observably in the fan's wing-blades and compartments; but the neutralization of the friction must not be expected to be an intelligible part of the two-fold taking-and-return procedure in the same way that two independent steps can be recognized and discriminated, by which the power itself is plainly dealt with in the operation, because although there are thus means to use power's dual shapes alternately, for self-compensation and economy no means are known by which the availability of driving-power can be recovered from its unavoidable wastes and losses here and there through friction. Although not actually a recurrent stream, in the sense of a whirling one with closed flux and reflux of actual and potential energy of circulation round a low-pressure core or vortex-axis, the idle-coursing current has yet a visibly reciprocating flux and reflux of its sensible and potential energy in its two-phased way of dealing with the driving power and at the same time also undiscernibly with the friction-checks opposed to it, which is just similar and equivalent in its rocking or reciprocating character between one of its two states of energy and the other, although not in any other respect identical in circumstances of its nature, with a fluid-vortex filament in continual orbitation. There is no idle-running companion air-stream through a Guibal fan, or through any kind of single-chamber fan, because these fans' constructions furnish no means for a taking-and-return transaction between such a current's energies and the driving power, on one side, nor, in consequence of this, can any part of the air-current through such a single-chambered fan free itself from frictional resistance on its route as a description of mechanical opposition which, when a fan's rate of revolution is uniform and constant, is directly and collectively confronted alone, or quite singly and indivisibly to the driving power.

Mr. T. A. Southern (Derby) wrote that he was very sceptical of the accuracy of the observations quoted in Mr. Capell's paper, or at any rate of their sufficiency as a basis for accurate and fair comparisons; nor did he agree with the writer's conclusions. He considered also that Mr. Capell's explanations of the discrepancies observed, by suggesting that fans (and the Capell fan in a special degree) had a strange and inexplicable knack of "throwing the water-gauge a long way back into the mine," was one which conveyed no clear idea, tended to confuse and

mystify the issue, and had no foundation in recognized laws relating to the motion of an elastic fluid, at uniform or varying velocities, in regular or irregular passages. Comparisons of ventilating appliances, if they are to be of any real value, must be based upon the averages of a long series of careful observations, in order to eliminate errors in the measurement of quantities, water-gauges, speeds, etc. In the last comparison given by Mr. Capell, viz., that between the fan and furnace at Silverhill colliery, he took the mean volume of air in the returns at 110,000 cubic feet per minute for the furnace, but this was not based upon any experiments, for he had shown* how these figures had been obtained, as follows:—"Mr. J. A. Longden, in a letter, gave the average quantity of air with the furnace at 1·4 inches water-gauge, as 110,000 cubic feet per minute. Mr. J. C. B. Hendy, in his tabulation (page 549, vol. ii.) in the *Transactions of the Federated Institution of Mining Engineers*, relating to the Capell fan at Silverhill colliery, puts the average work of the furnace there, with the same water-gauge, viz.:—1·4 inches, at 122,000 cubic feet per minute. But in a special measurement with 1·40 inches of water-gauge of the furnace (thermometer 52 degs. Fahr. on the surface, and 64 degs. Fahr. in the mine) he gave the writer as results 105,530 cubic feet of air per minute, measured in the returns. Therefore 110,000 feet may be taken as the mean result of the furnace ventilation with 1·4 inches of water-gauge on the doors below." Here is a comparison placed before the members, based upon the most haphazard data; an approximate statement by Mr. Longden, another by Mr. Hendy, one special measurement with a gauge of 1·40 inches, and then strike an average (as favourably as possible) and the thing is done. It is very difficult to read a water-gauge to within $\frac{1}{20}$ inch, but on this large volume of air $\frac{1}{20}$ inch error in the reading of the water-gauge would make a very great difference; consequently he thought that he was justified in saying, without any disrespect to the writer, that he had no faith in, and placed no reliance upon, any of the figures quoted by Mr. Capell.

Mr. J. S. Walker (Wigan) wrote that he had for a long time ceased to take the water-gauge at the fan-inlet; it was now his invariable practice to take it in the fan air-ways, preferably near the top of the upcast shaft. When this was done, unless there was some unusual friction in the pit, there was not much difference betwixt the water-gauge taken in this position and at the separation-doors at the bottom of the pit. With reference to the comparative tests of fans, he considered that in the case of two fans being tested upon one pit, unless designed for the same duty,

* *Trans. Fed. Inst.*, vol iii., page 198.

the comparative efficiencies of the types obtained would not be reliable. The writer wishes to refer the members to Mr. Barnes' statement* of the work performed by a Walker-Guibal fan at the Grassmoor collieries, and confirmed by Mr. C. E. Rhodes. The efficiency of the Guibal fan is now so well recognized, that it is not at all likely to be prejudiced by such comparisons as appear in Mr. Capell's paper. It would be instructive if Mr. Capell would inform the members, on what lines the fan at the Prosper I colliery, Westphalia, and that at the Coalpit Heath colliery, near Bristol, were constructed, and by whom?

Mr. M. Deacon (Alfreton) wrote respecting Mr. Capell's calculation that the Schiele fan at Shirland would have required not less than 7·55 inches of water-gauge in order to have produced 118,000 cubic feet of air per minute was theoretically correct, but it must be borne in mind that the fan was doing its maximum work at 1 inch water-gauge when it produced 43,699 cubic feet per minute. Although the experiments made upon the two fans were, in the writer's opinion, comparative at 1 inch of water-gauge, beyond that point he did not think that any calculation could be considered as fairly comparative. It would be interesting if Mr. Capell would describe the exact position of the orifice of the water-gauge tube and its direction in relation to the inlet of the fans in the Coalpit Heath experiments.

Mr. A. L. Steavenson (Durham) said no one had a greater admiration than he had for the way Mr. Capell had worked out his observations, and he would be pleased to see his efforts crowned with success. At the same time he agreed with some of Mr. Southern's observations as to the experiments being too fragmentary. What they wanted was a series of observations taken by persons whom they could trust. He did not mean to say they could not trust Mr. Capell, except in the respect that he had perhaps a little bias for the results of his own fan, naturally, and some people had accepted his conclusions. Mr. Capell said in his paper that "one point was prominent, and this was recorded at the time, viz., that there was a different water-gauge in the Guibal fan when passing the same volume of air as the small Capell fan;" this was a question of fact which should be thoroughly tested. A great deal depended on where the water-gauge was taken. He could understand that Mr. Capell would get a higher water-gauge at a distance from the fan than some of the smaller fans, such as the Schiele. Mr. Capell had said that his fan had six évasée chimneys in which it resembled the Guibal fan, but from the effects obtained he (Mr. Steavenson) rather doubted the resemblance. Unfor-

* *Trans. Fed. Inst.*, vol. ii., page 224.

tunately, they had only one case given in which they could test the manometric efficiency of the Guibal fan. Mr. Steavenson then referred to the results of numerous experiments in which the Guibal fan gave a manometric efficiency of at least 0·62. None of the open-running fans reached 0·50, simply because they had not an évasée chimney; if the Capell fan had six évasée chimneys the results should be very much higher than the 0·46 to 0·48 given as the average manometric efficiency. The paper only recorded the results of the trial of one Guibal fan of 16 feet diameter, giving 0·65, and here there was a manifest difference between the Guibal fan, which was 0·65 and the Capell fan which was only 0·48. The single case in which the Capell fan gave the manometric efficiency of 0·52 seemed to be wrong, as in none of the others did it attain 0·50. Hitherto it had been claimed that the Capell fan gave upwards of 68 and 70 per cent. of useful effect; now they had got down to something reasonable, and the Prosper I colliery fan was quoted at 55 per cent.; this was a very large reduction, but it was still an exceedingly good result, and he hoped the Capell fan would never yield less.

Mr. Capell said that result was obtained with 10·70 inches of water-gauge.

Mr. Steavenson, continuing, noticed that Mr. Capell had read another paper some time ago, but it was of so fragmentary a character that it was impossible to follow it.* They could not go into the question of volumes and water-gauges without a long series of accurate experiments. Mr. Capell had many volumes of experiments, but they were all taken in the same way and he feared they were not sufficiently reliable. He hoped, however, that before long they would have a Capell fan in this district, and have it thoroughly tested.

Mr. T. H. M. Stratton, following up the remarks of Mr. Steavenson, said the manometric efficiency of the Capell fan varied very largely. At the Prosper I colliery it was 0·50, at Gneisenau colliery 0·40, at Coalpit Heath colliery 0·43, and 0·50 at the higher speed, and at Silverhill colliery it was 0·43. There was a variation from 0·40 to 0·50, whereas in all other fans which he had tested the manometric efficiency was consistent and constant, in the Guibal being about 0·65, and in the Capell, Schiele, and Waddle, 0·46.

Mr. M. Walton Brown (Newcastle-upon-Tyne) said he presumed that Mr. Capell was alluding to the experiments upon the fans at Homerhill colliery when he said that "one point was prominent . . . . that

* *Trans. Fed. Inst.*, vol. iii., page 196.

there was a different water-gauge in the Guibal fan when passing the same volume of air as the small Capell fan." These experiments may be recalled, together with the 1867 experiments upon the same Guibal fan :—

Description of Fan.	No. of Experiment.	No. of Revolutions per Minute.	Volume of Air per Minute. Cubic Feet.	Water-Gauge. Inches.
Capell fan (5 feet) ...	1 ...	535 ...	14,128 ...	1·00
	2 ...	412 ...	11,171 ...	0·50
	3 ...	450 ...	21,600 ...	0·50
Guibal fan (16 feet) 1869	1 ...	26 ...	13,600 ...	0·15
	2 ...	65 ...	37,500 ...	1·02
	3 ...	96 ...	51,700 ...	1·75
„ 1885	1 ...	— ...	9,288 ...	0·50
	2 ...	82 ...	11,412 ...	1·00
	3 ...	93½ ...	14,508 ...	1·50

The 1885 experiments upon the Guibal fan might be discarded, and a comparison of the 1869 experiments showed that the Guibal fan produced a very much larger volume of air than the Capell fan running at the same water-gauge, viz., 37,500 cubic feet at 1·02 inches by the Guibal fan against 14,128 cubic feet at 1 inch by the Capell fan. The Homerhill and Shirland collieries' experiments proved conclusively that in each instance the fans were much too small and "inappropriate" to the resistance of the mines in question. Both were running under unfavourable conditions, and the resistances of the small fans to the passage of the air necessarily reduced the results. As to the action of the Capell fan, he (Mr. Brown) thought it was much the same as other fans of similar dimensions, producing a certain water-gauge at their inlets which was decreased if the readings were taken at increased distances from the fan. It was a recognized fact that a large fan would produce the same volume at a certain water-gauge when running at a lower speed than a small fan, thus :—

Description of Fan.	Diameter of Fan. Feet.	Revolutions of Fan per Minute.	Speed of Rim of Fan per Second. Feet.	Water-gauge. Inches.	Volume of Air of per Minute. Cubic Feet.
Guibal fan ...	30 ...	65·0 ...	102·1		
„ ...	16 ...	171·7 ...	143·8		
Waddle fan ...	30 ...	69·2 ...	108·7	2·883	100,000
„ ...	16 ..	184·1 ...	154·2		
Schiele fan ...	12 ...	224·2 ..	140·9		
„ ...	10 ...	322·2 ...	168·7		

He (Mr. Brown) would like information as to the mode in which the water-gauge readings at the bottom of the shafts, referred to in Mr. Capell's papers, were made; and also as to how a "fan of large exhausting capacity" can throw its water-gauge into the mine. If the water-gauge was correctly observed under precisely the same conditions at the

bottom of the upcast shaft leading to the ventilator, he could not understand how, if the water-gauge at bank was constant, that the water-gauge below ground was found to be less when one fan was running than with another; and all practical experiments coming under his observation support this view. He had made experiments, and found that the water-gauge at the bottom was practically constant under the same conditions. From the Appendix to the Report of the Prussian Fire-damp Commission (vol. v., pages 62 and 63), he found that the Guibal fan at the Prosper I collieries, No. 1 Pit, was 39 feet 4·45 inches in diameter at the tips of blades, and produced, at 50 revolutions per minute, or a blade-tip speed of 103·8 feet per second, 45,900 cubic feet of air per minute under 2·756 inches of water-gauge, the useful effect being only 0·202, at a manometrical efficiency of 0·58. Considerable improvement must have been made, and the resistances of the mine considerably decreased, in order that the same fan should produce, at a later date, 59,340 cubic feet of air, under 3·15 inches of water-gauge. It was very certain that an equal amount of improvement had been effected prior to the erection of the Capell fan, the equivalent orifice having been enlarged from 10 and 12 square feet for the Guibal fan to over 14 square feet for the Capell fan. He would also like to know the lines upon which this Guibal fan was built, and who was the maker, as there must be something to account for the low mechanical efficiency of 0·202.

Prof. H. Stroud (Newcastle-upon-Tyne) said that the given water-gauge did not indicate the true difference of pressure, as was evidently shown by examples taken from the paper. He should not imagine the same difference of pressure would give any different volume of air whatever the fan used, provided always that the circumstances were identical as to the resistance to air-flow in the mine.

Mr. A. L. Steavenson (Durham) said that, with respect to "throwing back," no doubt with different fans they would find a different vacuum at distances from the fan where they had insufficient area for the air to flow through, and he thought it was not a question of "throwing back" but of "swallowing" the volume. Of course the vacuum would be farther back with one fan than another, but "throwing the vacuum back" was rather an unhappy expression.

Mr. M. Walton Brown said with regard to taking the water-gauge at the bottom of the pit, it seemed to him that the correct way to do so was to carry a tube from bank. If the water-gauge was taken on the separation-doors the result would only be the difference of density on the two sides of the doors.

Mr. J. Hodgson (Edmondsley) asked whether Mr. Capell intended them to understand that one fan could exhaust more air than another at the same water-gauge: that is to say, suppose they obtained 50,000 cubic feet of air per minute with 1 inch of water-gauge at bank with the Guibal fan, would the Capell fan produce the same volume on the same mine at less than 1 inch of water-gauge; and were they to understand that they would get the same volume of air with a less water-gauge by the Capell fan?

The Rev. G. M. Capell (Passenham), replying to the discussion, first referred to Prof. Herschel's communication, which suggested a difficult line of thought. The nearest approach to it he could think of was its relation to the phenomena of the flow of the electric current. Referring to the more practical points raised he did not think any one present had been put in possession of a certain fact, viz., that the Capell fan, when it was run to produce initial gauge, closed off from the air altogether, gave a higher result than the Guibal fan. Mr. Howe in a paper before the Chesterfield Institute in 1871 stated that "the Staveley fans have been frequently run under different conditions for the sake of experiment, and whether the air is conducted through miles of pit roads, before reaching the fan, or simply exhausted from the atmosphere close to the inlet without passing through the pit at all, or if the air is effectually shut off from it and the ventilator turned round without any discharge, there does not appear to be any appreciable difference in the water-gauge indications for the same velocity of tips." The design of each Capell fan was different to any other, as he tried to apply each one to its special work. The following trials had recently been made upon the Guibal and Capell fans at the Marles collieries, in the north of France:—

	Guibal Fan.	Capell Fan.
Fan, diameter	23 feet. ...	12½ feet.
„ width	— ...	9 „
„ diameter of inlets ...	— ...	7 „
Engines, cylinders	— ...	2
„ diameter	— ...	26 inches.
„ stroke	— ...	52 „
„ fly-wheel, diameter	— ...	20 feet.
Revolutions per minute, engine	75 ...	61·5
„ „ fan	75 ...	305·0
Water-gauge, near fan ...	1·40 inches ...	9·60 inches.
„ in main drift ...	1·00 „ ...	9·20 „
„ in a side drift	— ...	9·00 „
Volume of air per minute	63,540 cubic feet. ...	180,000 cubic feet.
Horse-power in air	13·12 ...	272
Indicated horse-power of engine	32·23 ...	480
Mechanical efficiency	0·407 ...	0·578
Manometrical efficiency ...	0·348 ...	0·547

He advocated the static water-gauge as being the only one which would give true comparative results. There were certain conditions of a mine which they had not yet been able to find out the cause of, where neither one fan nor another would give its full useful effect.

Mr. A. L. Steavenson asked if the water-gauge had ever been tested on both the inner and outer cases?

Mr. Capell said they had, and the water-gauge taken upon the outer fan was always higher than that upon the inner fan. The object of the outer chamber was to decrease the resistance to the discharge of the air from the mine. They were all aware that there was no water-gauge visible when they had natural ventilation; it was not until they got past the speed of the natural ventilation that the water-gauge began to be visible. With regard to Mr. Southern's observations as to Silverhill colliery, he (Mr. Capell) could only say that he had never himself experimented in the fan-drifts or returns at this colliery. The results of the experiments were given to him by the two gentlemen as having been carefully taken. The water-gauge was fixed between the separation-doors in the pit, and in this position was common to the furnace and the Capell fan. There was no attempt at comparison, because the furnace observations were made just before the fan stopped, and he himself had had no opportunity of verifying the results, but the ventilation-books of the mine showed that the figures he arrived at, as the general results obtained by the furnace, were very fair ones. With regard to the improved condition of the resistance of the mine at Prosper I colliery, he could only say that the Prussian Fan Commission reported in 1884, and Mr. Herbst's report was made at the beginning of 1890; and it was quite possible that the conditions of 1890 were not those of 1884. But the Guibal fan was still on the mine and was run, alternately with the Capell fan, when it was first started in work. In using the words "throwing back," he only quoted from Mr. Deacon's paper. They would find in that paper that whereas, at the bottom of the downcast shaft, the water-gauge of the Capell fan, with 1 inch at the surface, was very slightly altered, that of the Schiele fan, at the same point below, was only 0·45 inch. He used the term to show that there was a fictitious gauge which did not affect the mine. If all experiments were made with the static gauge they would be able to compare one fan with another, and ascertain their actual ventilating powers on the mine. In reply to Mr. A. L. Steavenson's remarks that the Capell fan water-gauge was the same as the Waddle fan as to manometric efficiency, Mr. Capell cited the case of a 12 feet single inlet fan at Crespin colliery, near Anzin, giving 7 inches of water-gauge at 240 revolutions per minute, or

0·70 manometric efficiency. The 10 feet fan at Waleswood colliery produced 3·10 inches of water-gauge at 210 revolutions, or 0·574 of manometric efficiency. The 12 feet single inlet fan giving 4·75 inches of water-gauge at 225 revolutions, or 0·535 of manometric efficiency. The three Guibal fans tested by the Mechanical Ventilators Committee of the North of England Institute, in 1878, gave 0·52 to ·56 of manometric effect. The Waddle fan at Celynen colliery gave 0·457, and Schiele fans 0·32 to 0·333, of manometric effect. The Capell fans are designed to give certain manometric effects, and they nearly all vary in this respect, according to the condition of workings. In reply to Mr. Walton Brown's remarks about the experiments at Homer Hill colliery. The mine was not in the condition of the previous experiments referred to, as the return was blocked to increase friction, and the downcast shaft was covered with boards and sacking for the same object. In the early experiments, and in the tests with the Capell fan, Mr. James Swindell was the observer, and first noticed the gauge and volume discrepancies. With regard to the remarks about accuracy of observations, he (Mr. Capell) merely gave figures handed to him by others. The same class of discrepancy running, though results coming from unprejudiced sources, widely distant from each other, all point to some peculiarity in the fans' action, which he should be only too glad to have explained. A constantly recurring phenomenon cannot be explained away by a mere supposition that observations were wrongly taken. If only one or two cases existed, it might be so, but with large numbers some other cause must be sought. In reply to Mr. A. L. Steavenson, he (Mr. Capell) regretted that he used an inappropriate expression if it was so, but when he found that the water-gauge at 500 yards from the pit bottom was 1 inch with one fan, and 2 inches at the same spot with another, the gauge at bank being the same, the result was as if one fan actually threw back its gauge farther into the mine than another, the result, no doubt, of its greater capacity for swallowing air rapidly than the other fan. The observations of Mr. Howe on the Staveley Guibal fan, to the effect that the gauge was always the same at the same speed, showed that the gauge tube was in the inlet of the fan, where increased volume through increased air-velocity kept the gauge constant, whereas the increase of air into a large fan drift always lowered the gauge in the drift away from the influence of the inlet: gauge and volume being the two sides of a balance, if you increase the volume largely at a given speed the gauge is lowered and *vice versa*, if the gauge tube is not placed in a position to register the increased air-velocity.

The PRESIDENT said the discussion would be adjourned pending the publication of Mr. Capell's paper and the correspondence it had elicited. He would, however, move a vote of thanks in the meantime to Mr. Capell, to whom they were all deeply indebted for his attendance that day, and for the information he had given them, and although he had been pretty freely criticized he would no doubt feel that this had been done in a friendly spirit and with the view of getting at the facts of the case.

Mr. A. L. STEAVENSON seconded the vote of thanks, and said that it certainly would not be Mr. Capell's fault if the members had not benefited by the paper.

Mr. CAPELL acknowledged the vote of thanks, and said he would only be too glad to render any assistance he could with the view of arranging and carrying out experiments, and to give such time as he could spare to the matter.

The following papers were taken as read, and the meeting adjourned.

GOLD-MINING IN BRAZIL.

By E. M. TOUZEAU.

Introduction.

The industry of gold-mining in Brazil belongs more to the past than to the present, and consequently any attempt at description must contain much which appears to be out of date, and as such only of historic interest.

If, however, some connexion can be established between the methods pursued in the past and the present decadence of a once important pursuit such reference will be seen to be pertinent to the subject of this paper.

Before the great gold-fields of California and Australia had monopolized an attention now shared by South Africa, Brazil occupied the foremost position as a gold-producing country. The greatest returns were made during the last century and the first half of this; during the former period by native enterprise and during the latter principally by that of English companies.

The history of gold-mining seemingly repeats itself in each country nay, in each field, with such a fidelity to detail that the mistakes made have borne a general family resemblance, and that of Brazil forms no exception to the rule. A period when gold is found plentifully at surface or in shallow workings is followed by the lavish and often ill-advised expenditure of capital. To this succeeds a stage of discredit when sound concerns suffer with unsound, only to be terminated by a gradual weeding out of the bad, and a prosecution of the good ventures on legitimate lines.

Brazil reached the discredited stage and stopped. Her history therefore lacks the completeness afforded by that of later gold-producing countries which have each found their check to have been a salutary lesson, leaving them the stronger for its learning.

With the knowledge gained in more recent fields, the advantages of modern machinery, and the better means of transport now afforded, Brazil may yet hope to give a good account of herself, and emerge from her present obscurity. During the years which have elapsed since English

mining enterprise almost ceased in Brazil, the methods and means of extracting and treating ores have advanced sufficiently to make the poor and unworkable property of that date a workable and payable one to-day.

Immense quantities of the surface-clays, sand, and gravels, the latter called by the natives "cascalho," were excavated and carried to the "corregos" (streams) to be washed for gold. Hydraulicking was not attempted as now known, but reservoirs were built and the hills contoured for miles with "regos" (ditches) to convey the water from them, or the streams intercepted, to some favoured site for washing. At such sites, sluices called "canoas" were either excavated out of the natural rock or built of wood, and upon the rough bottoms of the former or in the hides or baize (bietta) stretched on the latter the gold was arrested. Between Morro de Santa Anna and Passagem many of these rock canoas may still be seen, and the whole surface of the country for a considerable area has evidently been excavated and washed in this way.

Washing in the rivers and streams also produced a large amount of gold, and even now it is very common to see a native, knee-deep in the water, raking up the bottom with an "enchada" (hoe), so that the disturbed dirt may flow through the little race he (or she) has made and over a few pieces of bietta upon which he (or she) may hope to arrest a few particles of gold as a reward of his (or her) day's work. The yield of the streams improves after heavy rains.

It was a common practice in the slavery days to send out individual slaves on such a mission, taxed to bring in a vintem of gold in a quill as the minimum of a day's work—the receipt for a smaller quantity being often given to them on their backs.

The writer has been told by the proprietor of one large fazenda that grains of gold are at times found in the crops of fowls killed on his property, and he has himself washed very fair samples from the alluvial mud and sand of river-banks, which shows that the streams still derive a supply from the waste of the surrounding country.

On one property visited by the writer gold to the value of £60,000 had been obtained from the outcrops of the reefs by the late Brazilian proprietor. Fires were lighted on the lodes, and when the rock became sufficiently heated cold water was thrown upon it. The rock thus broken was carried on mule-back about three quarters of a mile to stamps erected on the banks of a stream. The reefs are large, and must have been very rich at that time to cover so heavy a cost, and still yield such a handsome return.

At Ouro Preto, formerly called Villa Rica, innumerable small galleries were driven into the mountain-side, and small shafts sunk, from which an immense amount of mineral was extracted, and large quantities of gold obtained and, doubtless, also lost. For miles in an easterly direction the mountains are honeycombed by such shallow workings, and at Morro de Santa Anna 5,000 miners worked their separate claims or "datas" in a similar manner until the property passed finally into the hands of the Don Pedro North d'El Rey Mining Company, Limited.

The early Portuguese settler and his Brazilian descendants were wonderfully successful seekers of gold, but their mining knowledge was very limited; and water once reached, their works had to be abandoned.

In the early part of the present century English capital began to flow into Brazil, until at one time a number of properties were being worked and managed by English companies. Foremost amongst these was the Imperial Brazilian Mining Association, which will be again alluded to, the St. John d'El Rey, the Don Pedro North d'El Rey, and others.

They had great difficulties to contend with in bringing machinery and stores over long distances, in a country where, as yet, railways were unknown, and everything had to be conveyed in bullock-carts or on mule-back over an interminable series of the roughest mountain-roads.

Despite many drawbacks, mining was prosecuted and a very large amount of gold sent home, while the vigour of some of these undertakings is evinced by their continued existence to the present day.

This, the historical portion of the subject, might almost conclude the paper if its title were strictly adhered to, for little has been done in recent years to develop one of the greatest mineral districts of the world—a neglected and in part a forgotten one.

Mr. Henri Gorceix, late Director of the School of Mines at Ouro Preto, has fitly described the Province of Minas Geraes as having a "heart of gold in a breast of iron," and the simile only fails in its omission of reference to the wide range of mineral wealth which he himself has pointed out that it possesses.

Gold-bearing Formations.

Gold is found almost throughout the whole series of rocks in the gold-zone of Minas Geraes, but the bulk of that obtained by mining has been derived either from the quartz or jacotinga formations. The latter is a formation peculiar to Brazil which will be more especially alluded to later on. The quartz-formations have been extensively worked at Morro

Velho and Passagem, but with these exceptions no other property has been worked to any great depth or benefited by the use of modern machinery.

At Morro de Santa Anna, Rossa Grande, Pari, Antonia Pereira, and at many other points, operations have been conducted either by English or Brazilian companies, but in most cases under conditions which must have proved fatal to the success of almost any property.

The clay-formation has also been worked at several places with considerable success, notably at the Barra mine, situated near the village of Barra. This mine is the property of a Brazilian company and, as its name implies, the gold-bearing formation is mostly clay, yielding a steady and very remunerative return. It has been long and ably managed by Capt. Truran who works it entirely with native labour, and the rock requires most careful working to prevent accidents. The mineral is passed through Brazilian stamps driven by waterwheels. After stamping, the pulp is carried over blanket strakes and afterwards over tables having a good fall—a small quantity of water and a good fall being found to give the best results. The gold recovered is very fine and rarely visible in its matrix. The sand from the tables and blankets is washed by hand in bateas in the usual way.

With the exception of Passagem and Morro Velho the mineral has generally received the treatment described as used at the Barra mine, or in other words, a simple washing after stamping. It is reasonable to conclude that a very large amount of gold has been lost owing to the crude methods employed.

Method of Working.

The method of working has almost invariably consisted in following down a richly mineralized shoot in the lode, until water has been reached: then hand-power or, more rarely, that of water has been used to drain the excavation. Adits have been driven and shafts sunk; but anything in the nature of deadwork has been carefully avoided, and development regarded as a luxury too great to be indulged in. Small wooden pumps have generally marked the lowest level at which the Brazilian has cared to prosecute his search for the precious metal, and a portion of the free gold which the rock contained has satisfied him whether it would be worth his while to continue his work even above that horizon.

In most cases the mines were situated at points far distant from water-power and the mineral was carried on mule-back to the stream where water-power was available for crushing and treating the mineral; unless the ore was found sufficiently rich to bear this heavy cost the mine was abandoned.

The broken rock was often carried out of the mine by manual labour or at best wound by a mule or bullock-whim, and manual labour has been used even by some of the English companies for working the pumps.

The Brazilian stamps are almost entirely constructed of timber, having a wooden shaft with wrought-iron cams actuating iron tappets upon wooden stems. The shoes are of most excellent cast-iron made in the country, and the mortar boxes are of wood. If well constructed they may crush perhaps 15 cwts. per day, according to the nature of the rock; but it is more than doubtful whether the requisite degree of fineness is ever accurately determined. Where the ores crushed are pyritous, a heavy loss must occur by the simple method of gold extraction practised.

The Passagem mine, situated near to the old Portuguese city of Marianna, was formerly worked by the Anglo-Brazilian Gold Company, Limited, and at that time operations were conducted upon lines little better than those practised by the Brazilians. At present it is worked by the Ouro Preto Gold Mines of Brazil, Limited, and a very different state of things exists under the management of Mr. H. J. Gifford. The old incline-shafts have been enlarged and the mine developed upon modern principles. A rego of some miles in length brings an ample supply of water for power purposes and treatment of the ore. Hoisting and pumping, the latter a small item, is effected by water-power, which also drives a modern stamp-battery and a number of heads of Brazilian stamps.

The gold is rarely visible, and is mostly contained in pyrites, but the rock varies considerably. Specimens coated with asbestos have been taken from this mine. The treatment includes concentration upon blanket tables, pan-amalgamation, and a roasting and chlorination-plant. The latter was first most ably introduced and manufactured at the mine, and as it proved successful, a larger plant was sent out from England. The rock is trammed from the mine and then passes down chutes to the ore-bins, and from there to the rock-breakers and stamps. The whole operation is effected by gravitation and with a minimum of handling.

The Jacotinga Formation.

On account of its many peculiarities and the difficulty of working it, this formation possesses an interest apart from its being the only known example of gold occurring under such conditions. Roughly speaking it may be divided into two classes, the compact which requires stamping, and the soft which yields the bulk of the gold contained by the simple process of washing. These formations exist either separately or combined in a series of more or less parallel and irregular beds, striking

east-and-west with an underlie north or south of some 40 degs. They have been proved to be auriferous at various points for a distance of some 40 miles in a north-and-south direction, or at right angles to their strike.

The beds are not all auriferous, nor is any single one auriferous throughout its length; but shoots occur in groups at certain points parallel with the bedding and lying in its furrows. They dip irregularly towards the east about 30 degs., and contain gold unequally diffused throughout. In some shoots defined lines of gold or veins occur, which have been followed downwards at the Don Pedro mine for over 100 fathoms without any sign of their failing. Wherever the dip changes from a low to a high angle the shoots have become richer, and more or less they conform to the surface-contour.

The jacotinga is composed of micaceous iron ore, brown iron ore, saccharoid quartz, oxide of manganese, iron pyrites, iron glance, and clay, and in various shoots certain of these predominate or are absent, affording indications favourable or the reverse to the existence of gold.

At Morro d'Agoa Quente, Bananal, Brucutu, Pitangui, Sao Miguel, Itabira, Gongo Soco, Maquiné, and other points, English mining companies endeavoured in past years to work this formation with varying success. Of these companies the greatest and most successful were Gongo Soco and Maquiné, the former owned originally by the Imperial Brazilian Mining Association, the latter by the Don Pedro North d'El Rey Co., Limited. The only jacotinga mine now working is that at Maquiné.

At Gongo Soco, the lodes outcropped at the head of a valley, some 8 miles east of the town of Caethé. The dip of the auriferous shoots was easterly 8 to 30 degs., oblique to the normal direction and inclination of the beds. They were found to be very rich at this point, and were worked down to water-level, when the usual difficulties encountered made the owner willing, but not anxious, to sell. In 1825, the property passed into the hands of the English company, and from that date until 1856 mining operations upon an extensive scale were carried on. In 1832, the force employed by the company comprised 183 Europeans, 207 free Brazilians, and 404 slaves, the latter either owned or hired. During that year, 50,426 ounces troy of gold were obtained, 17,351·724 ounces being the result of stamping the compact jacotinga, and 33,074·988 ounces obtained by simply washing the soft variety.

Gold was found in this mine in extraordinary quantities at different times, and as much as 2,628 ounces were extracted in two days. A miner's hatcap-full taken out in January, 1829, upon being washed gave

331 ounces 14 dwts. of gold. Nuggets of over 240 ounces weight were met with, and in 1840 Capt. Blamey, assisted by two miners, extracted 1,260 ounces of gold in 3 hours.

The shoots dipped with the valley-line, and vertical shafts were sunk upon them at intervals; the deepest of these shafts was only 48 fathoms below adit. At this point difficulties were encountered, which led to the abandonment of the undertaking.

The treatment of the mineral was much the same as that to be described at the Don Pedro mine. The total amount of gold extracted from Gongo Soco mine between 1826 and 1856 amounted to 414,336 ounces troy.

The Don Pedro Mine.

The mine is distant about two miles north of the city of Marianna, and is situated in the "buraco" or "hollow" of Maquiné. The buraco occurs in a spur of the main range (see Figs. 1 and 2, Plate XIII.), and is partly the result of natural action and partly due to the removal of an immense quantity of material by the Antigos, who sluiced and open-cut very extensively here.

The capping of the spur, as also that of the main range, is an iron breccia, called canga. Under this lies a ferruginous sandstone, which forms the country rock, and enclosed in this are the beds of jacotinga, having an east-and-west strike, underlying about 40 degs. to 50 degs. north. In these beds occur the auriferous shoots before alluded to, dipping east into the spur.

Tradition assigns this place as the one where large quantities of gold had been extracted by one of the early Portuguese pioneers, who had taken a first instalment of his wealth home, and who died on his return journey without communicating the secret of the exact spot where he had made his find.

In 1865 a systematic search was undertaken by the Don Pedro North d'El Rey Co., which was then working the rock lodes of Morro de Santa Anna (another portion of their property), and they were rewarded by the re-discovery of the lost shoots. Great excitement followed the find, the rock mines were abandoned, and active operations were commenced at Maquiné. Adits were driven, and developments underground discovered shoots not known to exist at the surface, some branches being followed upwards, and pieces of gold taken out from under the grass roots.

The mine being at a considerable elevation, no difficulty was experienced with water, and gold was extracted in such quantities as enabled dividends of upwards of 100 per cent. per annum to be declared for three

years in succession, that is, 1867, 1868, and 1869. Deeper drainage then became necessary, and after several failures to drive adits through a belt of flucan which crossed the property, the existing adit, some 200 fathoms long, was completed. The water difficulty, however, was an increasing one, and though the pumping arrangements succeeded for some time, and allowed of a depth being attained of 50 fathoms on the incline, under adit, the machinery never had command of the water below the 30 fathoms level. Consequently, what with the breakdowns of expensive machinery, the re-opening of flooded workings in treacherous ground, and the difficulties attendant upon obtaining mineral under such adverse circumstances, the company had to suspend operations in 1880, and shortly afterwards the present company took over the property.

An interesting pamphlet, compiled from the official records of the company by Mr. John E. Dawson, contains fuller particulars of this mine than the scope of this paper will admit.

The present company has sunk an inclined shaft from the surface, 114 fathoms, and (to date) 59 fathoms below adit. The shaft is sunk in country rock, at an angle of 23 degs., and is 12 feet by 7 feet 6 inches within timbers. It is timbered throughout with the hardest and best timber procurable. The permanent sets are rarely under 20 inches through, and are spaced 6 feet apart, with intermediate sets between. Dividings, 10 inches by 6 inches, are placed under each cap, and the back and sides are close-lathed with candea poles. In some places where the ground sets off heavy on the timbers, rangers are carried under the caps, and the dividings are of iron. Double lines of tram rails of 28 inches gauge are laid in the whim end, and the engine end carries the column and pump-rods.

As in the formation itself, so in the country rock, great care is requisite in excavating the ground, which, often so hard as to require blasting, will in places run and bring excessive weight upon the timbers unless proper precautions are taken to avoid it. The sandstone is finely laminated, with intruding lines of jacotinga, which are always a source of danger.

This shaft had to be sunk diagonally under the old incline-shaft which crossed it between the 40 and 50 fathoms-levels, and over and under other levels, the work being successfully carried out without accident.

Cross-cuts are driven at intervals of 10 fathoms on the incline to the shoots. At present there are seven stopes carrying 9 feet caps with 9 feet legs, but these do not embrace the full width of the lode at this point, therefore stopes will be carried over these again until the whole formation is removed.

The system of stoping and timbering is shown in Fig. 3, Plate XIII., which shows a cross section of the stopes at the 50 fathoms horizon. The width of the lode has been proved at No. 4 stope by a rise to be 23 feet, but this does not represent its full width as the lode was still underfoot and water prevented any further test until such time as deeper drainage shall admit of its determination.

The workings have reached a point where the spur joins the main range, and it is here that predictions were made as far back as 1867 that the various shoots would unite into one main body of great richness. The shoots have formed the junction, and the rest of the forecast is being solved, so far with the most satisfactory results.

In the workings upon the formation the greatest care is required, and ordinary miners cause a series of runs which ruin the ground and are dangerous to life and the adjoining workings. The best men for this kind of work are natives trained from boyhood in the early operations of the company.

The back and side laths are of closely-fitted planks, tapered and driven together to prevent the slightest run, which must at once be stopped by ramming in "capim" (a kind of grass) at the back of them. Wet ends are kept closely breasted, and in driving, one breasting is heaved forth at a time and made secure.

The treacherous nature of the ground entirely disappears when the drainage of the formation is sufficiently deep, but for about 3 fathoms above water-level it continues saturated, and is, in its softer portions, almost unworkable, and decidedly so for profitable extraction.

The lines of gold have been referred to, but as they are one of the chief peculiarities of the jacotinga-formation further reference may not be out of place. They generally occur near to, but not in, the best looking jacotinga, and vary in value and size from a thread-like line to a rich "olha," or swell, jumping where a fissure occurs, sometime for many feet, but continuing in depth so far as yet attained.

The contortions of one of the auriferous shoots at the Maquiné mine is shown in Fig. 4, Plate XIII., and explains why an excavation 12 feet wide was required to remove the lodestuff. Fig. 5, Plate XIII., shows a cross section and the occurrence of another shoot, called the canoa or curve, from which large quantities of gold were extracted. The section also shows the position of the five lines of gold at that date.

To give some idea of the character of these lines the writer may mention his having seen about 5 cwts. of veinstuff taken out of one of them at the 45 fathoms horizon which yielded over 100 ounces of gold by

washing. Some of the prills were 3 inches to 4 inches long, and occurred as irregular slabs with sharp, jagged edges. A piece has been taken out of this mine 18 inches long, and on another occasion a bunch like seaweed, which fell to pieces on washing it. From pieces such as this the gold ranges down to the finest flour or float-gold, which, together with round shot-like pellets, require much care to prevent loss. The lines are carefully sealed with clay, marked by the mine captains with private marks, and guarded day and night till such time as the advance of the timbering permits of a further length being excavated. This is done in the presence of the head captain, who himself takes out the veinstuff and fills it into boxes, which are locked up by him and sent under guard to the reduction department. It should be mentioned that the various entrances to the mine are locked up at night, and all men are searched and have to change under inspection when lines are open in the mine. There has been both trouble and bloodshed in the past through gold-stealing, and life has been lost in consequence.

The mine is ventilated by means of an air-shaft parallel to and above the main shaft. From the bottom of the air-shaft wrought-iron air-pipes are led into the different workings, and an exhaust fan at the surface (driven off the main engine) keeps the mine as a rule well ventilated. In certain states of the atmosphere, however, the air is very dead, and what with the gases given off by the formation itself and the great masses of decaying timber in the mine, care has to be observed to avoid accidents

The mine being situated in a ravine sheltered from winds, but exposed to the full glare of a tropical sun, causes the question of ventilation to become occasionally inconveniently complicated and the temperature underground somewhat high.

The lowest parts of the mine remained under water from 1880 to 1890, when, the necessary machinery for draining it having been erected, the work of clearing out the old workings was commenced. These were found to have been filled up with the finest jacotinga carried in, and gradually deposited by the water until a mass had been formed, having the weight but not the stability even of the solid formation when wet. Great care had to be observed to avoid bursts of water, which, carrying with it the fine jacotinga, constantly threatened to fill the pumps up solid, and even with every caution would cut the buckets to pieces, sometimes after only half an hour's working.

The mine is now drained by means of a 5 feet Pelton waterwheel, situated upwards of 1,000 yards from the mouth of the shaft.

A line of 15 inches and 13½ inches diameter riveted steel pipes 1,530

feet long brings the water to the wheel under a head of 153 feet, and the power generated is transmitted by an endless wire-rope in two lengths to gearwork, situated near to the mouth of the shaft, which changes the rotary into a reciprocal motion. The speed of the transmission is upwards of 2,000 feet per minute, and the pulley-stands are of iron, four in number in the lower half and five in the upper half, carrying two sheaves 9 feet and 5 feet in diameter respectively, the upper sheave being 30 feet above the ground. The longest span in the transmission is 425 feet, and in this span the top or slack side of the rope is supported by an intermediate sheave. The gearwork actuates a line of rods working Cornish pumps with 15 inches plunger and 16 inches bucket lifts. The amount of incoming water is at present 26 cubic feet per minute.

The mineral is hauled to the surface by a mule whim, which is a slow and expensive method for over 100 fathoms of haulage; but a line of piping is laid and a second Pelton waterwheel will drive an air-compressor, the compressed air being led in pipes some 500 yards to an air-winding engine at the mouth of the shaft.

Until the new machinery can be completed for stamping, etc., the method of treating the ore will be that hitherto practised at this and other mines which have worked upon the jacotinga-formation. It is one capable of improvement, involves too much manual labour, not so much an object when slaves were employed, but important now that labour is free, scarce, and high priced.

Treatment of the Ore.

The ore is sent down to the reduction department either as "general" work or veinstuff, the latter oftener called "boxwork" on account of its being sent up out of the mine in locked boxes as before described.

The general work is trammed down an incline in cars controlled by a brake and tipped into the yard on the higher side of the reduction house, the boxwork being reserved for special treatment.

Separate wooden launders, carrying a stream of water regulated by stop-boards, run through the yard and discharge upon grizzleys and then upon flat copper gratings having slots $\frac{3}{8}$ inch by $\frac{1}{8}$ inch at the top end of each canoa. A woman is stationed at each launder to feed the ore into it regularly. Another native keeps each grizzley and grating clear of the "roughs," which are thrown aside for stamping. Prills of gold often come down in the general work and are picked off the gratings, a small reward being given to the native for each prill found. The sand which passes through the grating is carried by the water over an inclined table provided with riffles which catch the heavier gold.

From this table it falls into the head of a canoa or kind of long-tom having its narrow end lowest. The lower end is provided with stop-boards, which are gradually raised by another native who slowly rakes the rough sand as it accumulates, back against the water to the head of the canoa. A rich concentrate results, which is cropped from the head of the canoa for separate treatment with the boxwork.

The sand carried over the stop-boards at the lower end of the canoa then flows over blanket-strakes covered with a rough baize, the fine sand escaping and the rough being reserved for the stamps. The blankets are taken up at intervals by women who attend to them and these blankets are washed in a tank, the lowest being removed and replaced first. This sand is passed through a special canoa reserved for the veinstuff and croppings of the other canoas. Extra care is given to the rich mineral passed through this canoa which is slightly different in construction from the others, being smaller and having basket sieves of fine mesh. These can be lifted out for examination and the prills picked out. Small quantities only are passed and the whole operation is closely watched by the reduction officer to prevent theft.

The highly concentrated sand from the small canoa is washed by hand in bateas and the gold, almost clean, is put into a locked "taxo."

The tailings from the small canoa are caught and passed through the larger canoas with the general work so as to ensure, so far as the process will allow, a complete saving of the gold.

Jiggers were at one time employed, but their success was not sufficiently proved to warrant a further trial of them and the same may be said of buddles.

Great skill is shown by the expert user of the batea, and a single visible particle of gold will be shown up if placed as a test in a wash of dirt. Women are employed for this work in the reduction-house, and they use the batea with a circular sweep and side shake which brings the gold first to the centre and afterwards to the side of the batea as the waste is gradually washed away. To prevent the float-gold from escaping, the juice of a pounded herb is sprinkled upon the water in the batea when cleaning the gold.

A table is appended containing particulars of the amount of gold recovered and the percentage of loss by assay for several years. The loss, some 35 per cent., seemed excessive for free gold, and the writer sought a remedy in a finer crushing of the sand than had hitherto been adopted. The results obtained after five separate crushings and washings showed that gold of the utmost degree of fineness remained enclosed in the

extremely fine sand of the fourth crushing. Further attention to this matter will be given when the work now in hand permits of its being dealt with.

It will be seen from the table that the amount of gold extracted from the Don Pedro mine during the period from 1865 to 1892 amounted to 158,127 ounces troy, but it should be noted that mining operations were much retarded prior to 1880, in which year they stopped, and that even now they have not been fully resumed.

In concluding this paper, the writer would venture to suggest that the interest which attaches to gold-mining in Brazil is not so much due to what has been done, as to the possibilities of the country as a field for future work. Regarded in this light, it will be found of much greater interest and importance than the record of what has been, up to the present, a mere surface-scratching of a huge territory.

Year.	Weight Treated.		Gold Extracted.		Yield per Ton.			Loss by Assay of Waste.	Workmen Employed.		Working Costs.	Profit.	Loss.
	General.	Veinstuff.	General.	Veinstuff.	General.	Veinstuff.	Average.		European.	Native.	£ s. d.	£ s. d.	£ s. d.
	Tons.	Tons.	Oitavas.	Oitavas.	Oitavas	Oitavas.	Oitavas.	Per Cent.	No.	No.			
1865	3,137·8	...	14,949·5	...	...	...	4·35	...	...	...	...	...	...
1866	12,779	...	70,348	...	...	...	5·50	...	...	...	27,211 0 0	10,040 0 0	...
1867	20,153	...	182,397	...	...	...	8·91	...	23	345	26,137 19 8	54,572 0 0	...
1868	24,794·65	103·35	169,291	35,264	6·80	341·10	8·21	...	26	373	27,564 0 0	60,573 0 0	...
1869	25,472·40	81·60	168,253	32,114	6·60	393·55	7·84	...	33	418	32,370 0 0	55,428 0 0	...
1870	26,362·00	39·00	74,355	13,743	2·82	365·20	3·33	...	42	426	37,572 0 0	42 0 0	...
1871	23,825·10	144·90	95,564	26,331	4·01	112·69	5·08	...	39	379	41,954 0 0	8,606 0 0	...
1872	27,623·24	51·76	71,325	10,829	2·57	209·21	2·96	42·18	44	445	41,119 0 0	...	6,027 0 0
1873	30,448·50	14·19	58,109	6,170	1·908	434·81	2·144	35·00	46	427	40,297 0 0	...	8,316 0 0
1874	27,908	44·60	57,244	10,837	2·051	220·56	2·507	20·77	37	321	32,706 8 0	...	4,354 0 0
1875	24,583	...	7,830·2	...	3·18	...	3·18	...	...	...	31,703 0 0	3,280 0 0	...
1876	14,102	3·44	45,982	1,697	3·08	...	3·30	...	26	238	24,892 7 3	...	2,021 10 5
1877	22,686	...	51,601	35	2·30	...	2·30	...	26	290	28,366 11 8	...	4,994 0 0
1878	23,549	...	33,878	...	1·41	...	1·41	...	23	167	27,091 1 11	...	8,984 13 10
1879	19,031	...	10,588	...	0·55	...	0·55	...	20	152	...	...	19,759 0 0
1880	4,739	...	10,560	...	2·22	...	2·22	...	...	...	...	...	8,388 0 0
1881	884	...	2,320	...	2·62	...	2·62	...	...	...	6,571 19 11	...	8,749 0 0
1882	...	...	...	...	...	...	...	...	...	...	...	...	...
1883	...	...	...	...	...	...	...	...	...	...	...	...	...
1884	...	...	2,861	...	...	...	...	...	...	...	..	...	...
1885	...	...	3,614	...	...	...	...	...	...	...	...	...	...
1886	...	...	1,681	...	...	...	...	...	...	...	...	...	...
1887	...	...	...	...	...	...	...	...	...	...	...	...	...
1888	...	...	5,934	...	...	...	...	...	...	...	...	...	...
1889	...	...	2,220	...	...	...	...	...	74		...	...	...
1890	...	...	509	...	...	...	...	...	110		...	...	...
1891	...	...	4,434	...	...	...	...	...	145		...	...	...
1892	...	...	20,815	...	...	...	...	...			...	...	...

REMARKS.—1865, Boxes containing 96 lbs. of veinstuff, yielded upwards of 60 oitavas of gold. 1866, Line discovered yielding 670 oitavas per ton. 1867, One nugget extracted measured 18 in. by 3 in., and weighed 48 ounces; veinstuff extracted yielding 1,500 oitavas per ton. 1869, Water difficulty commenced. 1870, Difficulty continued; erection of heavy machinery commenced. 1871, Erection of heavy machinery continued. 1872, Erection of heavy machinery continued; lower levels under water. 1873, Erection of machinery continued, lower levels under water. 1874, Erection of machinery suspended. 1875, Erection of machinery suspended. 1876, Erection of machinery resumed. 1877, Erection of machinery resumed. 1878, Erection of machinery completed, but fails. 1879, Bottom of mine under water. 1880, Additional machinery erected. 1881, Failure of machinery; suspension of operations. 1883, Property passed into hands of Don Pedro Gold-Mining Co. 1884, Sinking of new shaft commenced. 1885, Sinking of new shaft continued. 1886, Sinking of new shaft continued. 1887, Mine drained to 40 fathoms level. 1883, Appliances fail to deal with accumulated water. 1890, New machinery erected and mine drained; and re-opening bottom of mine. 1891, Sinking new shaft to 50 fathoms level. 1892, Sinking new shaft to 60 fathoms level.

THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.

By A. G. CHARLETON.

Part I.—Introduction.

The title of this paper has been suggested to the writer by a most interesting contribution on the subject of gold-quartz reduction, read by Mr. A. H. Curtis, at a recent meeting of the Institution of Civil Engineers, which the writer in common with other visitors, had the pleasure of hearing discussed at the two following sessions.

The subject is of such wide scope, that the writer ventures to think some of the points Mr. Curtis dealt with may be enlarged upon, and in fact the subject can be approached from several different points of view (notably from a metallurgical one) without exhausting it, or in any way attempting to trench on matters which Mr. Curtis has more particularly disposed of.

In summing up, Mr. Curtis arrives at certain general conclusions, which are almost entirely in accordance with the writer's own experience, viz.:—

I.—That the most suitable machines for the preliminary coarse-crushing of nearly all ores (depending on individual circumstances, referred to later on) come under one of the two following classes :—

A. Those with a reciprocating-jaw action, either hung at the top on a rocker shaft, so as to swing freely backwards and forwards at the bottom (on the well-known Blake and Marsden principles), or pivoted at the lower end, so as to move to the greatest extent at the top (like the Dodge).

B. Those with a gyratory movement (like the Gates or Comet crusher).

II.—That, for medium-coarse (below $2\frac{1}{2}$ inches) to fine-crushing, necessary for an ore which has to be amalgamated, concentrated, lixiviated, chlorinated, or treated by any of the various modifications of these processes which have sprung up of late years (each with some special claims of its own), the types in their own particular provinces, of the best classes of machinery to employ, are represented by :—

C. Gravitation stamps.

D. First-motion rolls for medium and fine work ; or geared-rolls for coarser ore sizes, and

E. Centrifugal roller-mills (such as the Huntington) which have stood the test of wide use, and time.

If, however, it is intended to reduce to pulp ore already crushed fine, as required in certain branches of amalgamation, one must add to this list:—

F. Edge-roller-mills of the improved Chilian type, Frankfort mills; and what are known in America as

G. Combination-pans, a modified form of the old Wheeler-pan.

Coarse-crushing.

Although hand-spalling with a sledge still holds its own for breaking gold-quartz in a number of Queensland mills and other places where stereotyped prejudice still holds it in position, mechanical means should generally be used for breaking the ore (as it comes from the mine) to a convenient size for its subsequent treatment. The exceptions to this rule are: If the stone contains mineral, which can be picked out at once and shipped to smelting-works to be otherwise dealt with; if manual labour be exceptionally efficient and cheap; if power of any sort be exceptionally dear; if the location of the mine* render the transport and operation of heavy machinery practically out of the question; or if the location of an existing mill of old construction will not allow of the introduction of an ore-breaker with its accompanying bins, grizzleys and automatic-feeders, etc. In point of fact the first is the only case in which spalling can compete with rock-breakers in a modern mill properly located and arranged,† as for instance, where a concentrating ore carries some such combination as argentiferous galena and zinc blende, mixed with carbonate of lead, and copper pyrites or iron pyrites, spathic iron and quartz. The result of crushing ore like this, would be to break up much of the blende, pyrites, etc., into grains, which, with specific gravity ranging only from 3·9 to 5·1, could not be thoroughly separated from one another by jigging, or any other mechanical dressing process. A mixture of these minerals‡ has no market, and hence crushing without some previous selection would obviously result in loss.

The ore for example at Ems which is divided into (1) rich galena ore; (2) cobbing ore; (3) pyritiferous cobbing ore; and (4) barren rock furnishes an illustration given by Kunhardt.§

In many localities, however, as he remarks, loss or no loss, the value of labour is too high to admit of spalling and cobbing being carried out together (as they have to be in such cases) and at best it is a pain-

* Since the introduction of sectionalized machinery and the application of electricity to long-distance transmission, remoteness from civilization influences this aspect of the question, more than local engineering obstacles.

† Assuming it to have a capacity of 20 tons a day, or anything over that amount.

‡ A bove certain proportions small enough to be neglected.

§ *School of Mines Quarterly*, series 2. "The Art of Ore Dressing in Europe."

fully primitive practice, which, carried out on a large scale, retards or even prevents, by the sedentary nature of the work, a healthy physical development of the youth of the mining population, though on the other hand, it certainly tends to stimulate their faculties of observation.

In class A, the best-known crushers are the Blake and Blake-Marsden stone-breakers.

The Blake-Marsden hand-hammer-action crusher takes, it is claimed, less power to get through a given amount of work than the Blake; nevertheless it does not appear to have come into such general favour for mining purposes. Perhaps this may be accounted for by its greater prime cost, weight, and the large space it occupies, for it seems a simple and well-built machine.

The chief disadvantages, which attach to the use of the above class of ore-breakers, are :—

(*a*) That the moving jaw only does effective work during the forward stroke, rendering the action an intermittent one.

(*b*) That thin, flat pieces of rock may at times pass the jaws without being broken, an objection, it may be remarked, which is not of very great consequence to the miner, unless the product of the machine is unusually large, and goes direct to automatic feeders.

(*c*) That the stone leaving the machines is of continually varying sizes.

With regard to the Dodge, its special advantages are :—

(*a*) Its light weight.

(*b*) The jaw being fixed at the bottom, (1) the size of the product can be more easily regulated; (2) is more uniform in size; and (3) the stone can be crushed down finer than with crushers of the preceding class.

The Dodge is more particularly recommended on these grounds, for fine-crushing in concentration-works, although its capacity is more or less limited as compared with the Blake, owing to the fact that it gives a finer product.

It should also meet a want in small mills, which cannot keep a more powerful breaker fully employed, or as a preparatory machine for finer crushing, in Rolls or a Huntington.

Belonging to class B there are :—

1. *The Gates or Comet Rock and Ore-breaker.*—Owing to the gyratory movement of the breaking-head of this machine on an eccentric, within a circular outer shell, which is protected with toothed liners, and is shaped like an inverted cone, it is claimed :—

(*a*) That flat pieces of rock cannot pass through it uncrushed to the same extent, as in a machine with reciprocating-jaws.

(*b*) That as the area of breaking-surface increases towards the discharge end, the material as it is broken smaller is spread over a larger space, which results in finer crushing than is practicable with a reciprocating jaw-machine.

(*c*) That as the head is loose on the eccentric spindle, there is no grinding action, and there is a line of crushing-surface from top to bottom of the head and liners in constant action.

(*d*) The area of breaking-surface is about three times as great as in reciprocating machines of similar power. This seems to admit of the machine doing nearly treble the work in any given time, driven at the same speed, and of tipping into it large quantities of stone (using a hopper); dispensing with much of the attention necessary with machines belonging to class A.

(*e*) That, having a comparatively small crushing-surface acting continuously, dispensing with a fly-wheel to store up power (part of which is wasted), and the rock which is to be broken being supported at its ends only, in the circular crushing-chamber, less power is required, as compared with a jaw machine, to do a given amount of work.

Quoting the report of a committee, on a competitive trial between a 9 inches by 15 inches Blake-Marsden, and a 9 inches by 14 inches Gates, at Meriden, Connecticut, on May 30th, 1883, the makers of the Gates assert that, crushing the same kind of stone to an equal size, the latter machine crushed over three times as much in a given time, and showed a saving of about 33 per cent. of the indicated power consumed in doing the work.

On the other side, it must be said, that a Gates of an equal capacity will not take in a lump of rock of as large sectional dimensions as a Blake or Marsden; and the latter machines have the advantage in respect of weight and prime cost; the price of the large sizes is in fact extremely heavy, and the height of the Gates is also a disadvantage, where fall has to be economized. The injury done to the machine, which might result from a hammer head, or drill falling into it (a not unfrequent occurrence through carelessness, or occasionally other reasons) has also to be considered, and there is no doubt that the Gates requires particular attention, in keeping it properly lubricated with suitable lard, or heavy lubricating oil. The head and liners are made of specially chilled, cast white-iron.

There is one particular case, however, which should be pointed out, in which the Gates seems to possess very special advantages over other crushers, and that is, where there is a wide lode yielding large quantities

of milling ore, in which the stone breaks "large" in stoping it down, and requires in consequence to be spalled, to get it into the jaws of an ordinary breaker. Under such circumstances, a Gates may obviously serve a very useful purpose indeed, as the larger sizes will take in whole lumps, as big as can be handled, and by this means the tedious and expensive process of breaking in the mine, or "breaker-floor," is avoided.

The measure of economy in using a large sized breaker of the kind will necessarily depend, however, on being able to keep it fully employed, as well as possessing ample storage-bin room and driving power. As an instance of the work it can accomplish under such circumstances, the Caledonia mill in Dakota may be quoted,* where one No. 6 Gates, with receiving-openings 12 inches by 18 inches, attended by one man, crushed 200 tons in 10 hours, and did the work formerly set for three No. 5 Blakes (the largest ordinary pattern in the market), with a receiving-opening 9 inches by 15 inches. These latter took five men 20 hours, it is said, to produce the same result, requiring the same amount of power. It is asserted by the makers of the Gates, that its action entails less stress on the structure it is mounted upon, than is the case with a reciprocating breaker, but one is somewhat inclined to doubt whether this holds good in starting it when empty, whatever may be the case whilst it is running full.

Its size, and weight, must evidently be special objections, when it has to be mounted on a lofty structure (like the back floor of a stamp mill); and where the stone is divided, along a long line of bins, the advantage of using one large machine, is more or less offset by the difficulty of distributing the ore between them, unless there is an exceptionally large fall to spare. Under such circumstances the only practical escape from the difficulty, is to set the breaker apart from the mill (if possible close to the shaft); and it is not improbable that this is, as a rule, (where the mine and mill are owned by the same company, and the former keeps the latter entirely occupied), the best disposition to adopt. This does away with much of the dust, which plays havoc with the bearings of the other machinery of the mill, permits of lighter framing for foundations and superstructure, less grading (in some cases),† and a smaller annexe to the main building.

2.—*The Lowry Ore-breaker.*—A second form of gyratory crusher has quite recently been introduced by the National Machinery Co., of Tiffin, Ohio.

* "Gold-milling in the Black Hills," by H. O. Hofman. *Trans. Am. Inst. Min. E.*, vol. xvii., page 498.

† When rock-breakers and ore-bins are used with automatic-feeders at the back of a stamp-battery, which has tables and concentrators below it, the least practicable fall that ensures the automatic movement of the ore through the various stages of the process is about 33 feet from the rock-breaker to the concentrator floor.

A description of this crusher appeared in the *New York Mining Journal* of December 5th, 1891.

Its construction is illustrated by a diagram, which shows it in section, with a removable revolving top for fine-crushing, containing two or more holes, to regulate and spread the stone round the breaker-head. The ore is thrown into a hopper, and is discharged as fast as it is broken, avoiding any danger of clogging. The Lowry ore-breaker appears to occupy the same position with regard to the Gates, that the Dodge does to the Blake, as the vertical shaft is fulcrumed below the crusher-head instead of above it; consequently there is less motion, so it is asserted, at the bottom of the jaw, and the product is said to be more even than that of any of the jaw-crushers, in which the jaw is pivoted at the top, a statement which is no doubt correct.

The chilled-iron head or cone appears smaller in the Lowry breaker than in the Gates. The chilled-iron ring or liner, which forms the outer crushing-face is cast in one solid piece, having a solid turned bearing and supporting-backing behind it. The machine is adjusted for crushing to various sizes by screws at the side of the crushing mortar, by which the liner can be raised or lowered. This is done to obviate the necessity of lifting the vertical shaft, carrying the crusher-head, the bottom of which always remains in the oil chamber of the eccentric, at the base of the machine. The vertical shaft is driven by ropes with sufficient laps round the pulleys, and with provision for adjustment in the tension-pulley.

As one of the chief advantages of this machine, the manufacturers claim that the crusher-head strikes downward, and has a tendency to draw in the material to be crushed, which would not be the case if the vertical shaft were fulcrumed at the top.

These breakers are made in nine sizes, having a capacity varying between 5 and 250 tons per hour.

Without having seen both machines at work and knowing more about the Lowry, it is impossible to institute a fair comparison between it and the Gates. But it is probable that, comparing the former with reciprocating-jaw machines, most of the arguments for and against the Gates seem to apply equally to its new competitor.

The purpose for which an ore-breaker is intended, its capacity, saving in space occupied, weight, quickness in setting to work, ease of repair, simplicity of construction, wear per ton of ore, liability to serious or minor accidents, first cost, and consumption of labour in attendance, lubricants, and power required per ton of ore crushed, will all influence the choice of a machine, and no hard-and-fast line can be drawn, between the

various coarse-crushers which have been mentioned, each possessing a sphere of usefulness of its own. In fact, as some of the above conditions are to a certain extent conflicting, a careful consideration of the particular circumstances of each case is absolutely necessary. In judging of the relative economy of different crushers in regard to consumption of power, it is perfectly correct that the only true test of their efficiency is the actual, indicated, useful effect shown per lb. of fuel burnt in cubing a certain class of stone to a given size. This method of comparison is, not however, so entirely reliable as would at first sight appear, unless the machines under trial be driven with the same engines and boilers, by the same enginemen.

The consumption of fuel will, in fact, differ in different works, according to its quality or description, the size and type of engine and boiler employed, and the care bestowed in looking after them.

Further than this, it is absolutely necessary to compare the exact time the crushing lasted, deducting all stoppages, since an engine which is idle, during the time the boiler fire is alight, may be consuming—in proportion to the useful work done—much more fuel than either the indicated or real horse-power (measured while the engine was working) would lead one to suppose. The different sizes, weights, capacities, size to which the stone is cubed, size and speed of driving-pulleys, nominal horse-power expended, and cost of the Blake, Blake-Marsden, Dodge, Gates, and Lowry ore-crushers, are given in the subjoined tables, as far as it is possible to ascertain them.

DIMENSIONS, PRICES, AND PRODUCT OF THE BLAKE STONE- AND ORE-BREAKER.*

No.†	Size of Receiving Opening.	Approximate Product per Hour in Tons crushed to 2 inch size	Weight of Heaviest Piece.	Total Weight.	Extreme Dimensions.						Driving Pulley.				Proper Speed. Revs. per Min.	H.P. Required.	Price F.O.B. Chicago.		
					Length.		Breadth.		Height.		Diam.		Face.						
	Inches.		Lbs.	Lbs.	Ft.	In.	Ft.	In.	Ft.	In.	Ft.	In.	Ft.	In.			£	s.	d.
...	3 × 1½	Laboratory	40	100	1	1	0	6	0	10	0	5	0	1	300	...	10	8	4
A	10 × 4	4	1,993	4,719	4	6	4	0	3	11	1	8	0	6	...	4	62	10	0
2	10 × 7	6⅔	3,950	7,896	5	4	4	6	4	5	2	0	0	7½	275	7	104	3	4
5	15 × 9	10⅔	7,179	15,749	6	8	5	0	5	3	2	6	0	9	...	10	156	5	0
8	20 × 10	13⅓	7,700	17,000	6	10	5	9	5	11	3	0	1	0	250	14	218	15	0

* Messrs. Fraser and Chalmers' *Catalogue, No. 4* (Gold and Silver Mills), and Egleston's *Silver*, page 358.

† There are various intermediate sizes not mentioned in the table; the 20 inches by 15 inches, with a product of 20 tons per hour, being the largest.

Size A—Takes pieces 4 inches thick, and crushes to ½ inch or less. 2—Takes pieces 7 inches thick, and crushes to ¾ inch or less. 3 and 4 are used for road-ballast, furnaces, and breaking smaller for other crushers. Mr. H. S. Munroe* places the cost of repairs for eight 7 inches by 15 inches lever-pattern crushers (Blakes) with corrugated jaws, estimated to have crushed 224,203 tons of limestone, at Bonne Terre, Missouri, in 12 months, at:—

	£	s.	d.
12 levers, at $25·00	62	10	0
9 jaw-plates, at $15·00	29	1	3
12 jaw-plates, at $12·00	30	0	0
Toggles, check-plates, and sundries ...	51	12	6
Total cost	£173	3	9

This represents an average of about £20 16s. 8d. for each crusher, breaking 80 tons per diem, to 1½ inches, but does not include the cost of babbitting bearings, or labour in making repairs.

From these data, the average life of the wearing parts of a jaw-crusher, appears (similarly circumstanced) to be about 8 months.

The lower box of the pitman-shaft must be packed with thin wood packing, to prevent the key tightening the box to the shaft. Iron plugs should be kept in the oil-holes, and if the fixed jaw requires to be cast-up, it should be backed with zinc plate about ¼ inch thick.

The amount of product depends on the distance the jaws are set apart, the speed, and also on the nature of the ore.

The product given in the table assumes the jaws set 1½ inches open at the bottom, the machine run at proper speed, and properly fed, but it will vary somewhat with the character of the stone, brittle stone for instance going through faster than sandstone.

The No. 2 Blake is made in sections for mountain transport. Total weight, 6,881 lbs.; cost, £145 16s. 8d. No piece of this machine weighs over 332 lbs.

DIMENSIONS, PRICES, AND PRODUCT OF THE DODGE CRUSHER.†

No.	Size of Jaw Opening.	Approximate Product per Hour to Nut Size.	Total Weight.	Driving Pulley. Diameter.	Width of Belt used.	Proper Speed. Revolutions per Minute.	H.P. Required.	Price F.O.B. Chicago, Complete.
	Inches.	Tons.	Lbs.	Inches.	Inches.			£ s. d.
1	4 × 6	½—1	1,200	20	4	275	2—4	52 1 8
2	7 × 9	1—3	4,300	24	5	235	4—8	93 15 0
3	8 × 12	2—5	5,600	30	6	220	8—12	114 11 8
4	10 × 16	5—8	12,000	36	8	200	12—18	187 10 0

* "The New Dressing Works of the St. Joseph Lead Company." *Trans. Am. Inst. Min. E.*, vol. xvii., page 659.

† Messrs. Fraser and Chalmers' *Catalogue, No. 4.*

DIMENSIONS, PRICES, AND PRODUCT OF THE BLAKE-MARSDEN (WITH LEVER AND ECCENTRIC MOTION) STONE-BREAKERS.*

Size of Receiving Opening.	Approximate Product per Hour to Road Metal Size, 2½ Inches.	Total Weight of Machine.	Extreme Dimensions.			Driving Pulley.		Proper Speed. Revolutions per Minute.	H.P. Required.	Price.
			Length.	Breadth.	Height.	Diam.	Face.			
Inches.	Tons. Cwts.	Lbs.	Ft. In.	Ft. In.	Ft. In.	Ft. In.	Ft. In.			£
12 × 8	5 8	10,080	8 1	3 7	4 9½	2 0	0 5	250 to 300 revolutions.	5	113
15 × 8	6 5	12,880	8 9½	4 10	5 3	2 6	0 6		7	135
15 × 10	7 10	15,680	8 9½	4 10	5 3	2 6	0 7		8	150
18 × 9	8 15	17,584	8 11	4 11	5 0½	3 0	0 7		9	165
20 × 10	10 0	22,400	9 11	5 5	5 4½	3 0	0 7		10	180
24 × 13	15 0	30,240	10 6½	5 8½	6 7½	3 0	0 9		12	263
24 × 16	16 0	31,360	10 6½	5 8½	6 7½	3 0	0 9		14	275
24 × 17	16 5	31,920	10 6½	5 8½	6 7½	3 0	0 9		14	282
30 × 18	20 6	33,600	...	...	...	...	...		18	375
34 × 18	23 2	42,560	...	...	...	...	...		20	413

The makers state that it is advisable to use an engine of 2 horse-power more than stated in the table, as it is more economical to run a larger engine, than to run one up to its full power (this remark also applies to the Blake and the Gates crushers). The principal working parts are all crucible steel.

DIMENSIONS, PRICES, AND PRODUCT OF THE GATES OR COMET ORE-CRUSHER.

Size.	Dimensions of the Three Receiving Openings combined.	Dimensions of each Receiving Opening, about.	Weight of Breaker.	Capacity per Hour in Tons of 2,000 Lbs. passing 2½ inches Ring according to Character of Ore.	Dimensions of Driving Pulley.		Space occupied by Breaker set up.			Diameter of Hopper.	Revolutions of Pulley.	H.P. of Engine recommended to drive Breaker, Elevator, and Screen.		Net Price F.O.B. London.
					Diam.	Face.	Height from Bottom Frame to Top of Hopper.	Width of Frame.	Length of Frame.			Limestone.	Granite.	
	Inches.	Inches.	Lbs.		In.	In.	In.	In.	In.	In.				£
00	2 × 12	2 × 4	500	{Laboratory and sampling}	8	2⅝	24	17	26	13	700	½	½	25
0	4 × 30	4 × 10	3,100	2—4	16	6	48	30	73	28	500	4	4	85
1	5 × 36	5 × 12	5,500	4—8	20	7	54	31	76	37½	475	8	8	125
2	6 × 42	6 × 14	7,800	6—12	24	8	60	39	90	39½	450	12	15	165
3	7 × 45	7 × 15	13,500	10—20	28	10	73	48	103	44⅓	425	20	30	250
4	8 × 54	8 × 18	20,000	15—30	32	12	85	54	114	51	400	30	40	400
5	10 × 60	10 × 20	27,000	25—40	36	14	96	63	123	59	375	40	50	500
6	11 × 72	11 × 24	36,000	30—60	40	16	109	73	139	66	350	50	60	725
7½	13 × 90	13 × 30	60,000	40—75	48	18	116	73	144	120	350	60	75	1,040
8	18 × 135	18 × 45	89,000	100—150	48	20	156	90	164	132	350	125	150	1,450

* Makers' catalogue and correspondence.

The power required per ton will, of course, vary with the size to which the stone is broken, requiring additional power, to break it smaller than ordinary macadam.

An elevator and screen are used when it is desired to break everything down to $\frac{3}{8}$ inch.

If the product is not required smaller than $\frac{3}{4}$ inch, and some "spalls" are not objectionable, the screen and elevator are not needed.

Prices of Gates' breakers with revolving screen and return-elevator fitted complete, on frame; size 0, with 24 inches by 36 inches revolving screen, £156; size 1, with 24 inches by 36 inches revolving screen, £208; size 2, with 24 inches by 36 inches revolving screen, £250; size 3, with 24 inches by 36 inches revolving screen, £354.

Elevators 24 feet between centres, including connexions, range in price for the various size breakers (from 1 to 7) from £56 to £127, and weigh from 2,000 to 5,000 lbs. Revolving screens, with dust jackets, cost from £42 to £146 (exclusive of connexions) and weigh 2,000 to 8,000 lbs.

Medium-coarse and Fine-crushing.

From the point of view of the mechanical engineer the superiority of one machine of class II. over another, for the reduction of a certain kind of stone (say granite), can in a measure, be pretty easily demonstrated, but the question as a miner requires to look at it, becomes one of extreme complexity, as the factors to be dealt with are variable and conflicting, in more ways than one.

Viewed in this light, if the field of selection be limited, merely to the three classes of crushers particularized on page 87 the choice of the best for any special case, appears to depend on three equally important considerations, which are altogether outside the primary question of the mechanical merits or demerits of a particular machine, and it is here that the experience of the mining engineer is called into play.

To arrive at a trustworthy solution of the problem, he must take into consideration:—1st, the character of the ore itself and its associated gangue; 2nd, the degree of comminution to which it is desirable to carry reduction in order to get the best results out of any particular system of treatment that may be adopted; whilst in some cases, the location of the mine and the nature and position of the mill site, also form a 3rd factor of the question.

To put the matter in a nut shell and go a step farther, it may be laid down as a rule, that setting aside any other machines which have claims to attention, where one of the three classes of crushers (stamps, rolls, and roller-mills) now under consideration is found, commercially speaking, to best meet the needs of the miner in any one instance, it is most unlikely that either of the other two types, will be found suited to the requirements of the same case ; if indeed, they did not turn out on trial absolute failures, supposing one kind of machine to be substituted for another.

An ideal "reduction plant," would have to establish a claim to extract the largest percentage possible of gold or other metal automatically in the shortest possible time, and in the simplest possible manner, combining capacity for expeditiously reducing the greatest quantity of ore in a given time to any desired point of fineness, with a minimum production of fines or slimes; it should further possess strength and durability without being cumbersome, and have as few working-parts and auxiliary appliances connected with it as possible. Now, while this acme of mechanical and metallurgical perfection combined has yet to be invented, the success of one type of crusher as against another in various instances, may be directly traced to the fact that some of those features which are wanting in one class of machine, are found more or less developed in one or other of its competitors. In selecting the most suitable machinery for his own particular object it is therefore incumbent on the miner not to lose sight of the points that have been indicated, as they may relatively affect in a different degree the circumstances he happens to have to deal with.

The adaptability of the plant in the foregoing respects, however, is not everything, as the outlay it may involve must after all be justified by the returns it will yield. He has therefore to compare :—

1. The prime cost and weight of each class of machinery, with its indispensable adjuncts for extracting the commercially valuable metals from any particular ore, representing a very variable capital outlay to produce a given result.
2. The simplicity or complexity of the entire plant in each case. not only with regard to cost of repairs, but also the time lost in maintaining it in running order in out-of-the-way places. difficult of communication, and where skilled labour is perhaps not available.
3. The relative time required to erect each different class of machinery ; and lastly,

4. The comparative costs (both in regard to quantity and price) of labour, fuel, oil, etc., required to run a given class of plant in a certain locality.

Under these circumstances, with the limited data possessed at present in the shape of independent figures throwing light on many of these questions of relative efficiency and cost, at most only general statements can be made; and any attempt to assert the adaptability or superiority of one machine or class of machines for all cases, must infallibly break down.

Each form of crushing machinery has its champions and detractors who freely express themselves for or against it, according to the measure of success it happens to have achieved merely in their own limited personal experience; and where failure occurs the machine not unfrequently is blamed, when in some instances it is owing to want of forethought on the part of the purchaser in overlooking natural contingent results, which might have been reasonably foreseen and easily escaped.

The degree of comminution to which it is desirable to carry reduction to get the best results out of any particular system of treatment rests on the general nature of the process employed to extract whatever of value the ore contains; this is dependent to a very large extent on the ore itself and its associated gangue.

The choice of medium and fine-crushing machinery must then be regulated, from a mechanical standpoint, as well as by the class of ore it is required to crush, bearing in mind that ruinous losses may ensue by going to either extreme. That is to say, in selecting a machine, if crushing is carried to an unnecessary degree of fineness, money is wasted in doing unnecessary work, which shows itself in the cost sheet at the end of the year, besides risking various mechanical and metallurgical losses.

Amongst these latter, may be enumerated, slimes and fine mineral in suspension, in concentration, and wet amalgamation; dust in furnace and other dry processes; and various inconveniences in filtering and extraction, in those cases where ore has to be lixiviated; all entailing extra expense, to mitigate a difficulty which has been artificially created.

If, on the other hand, the crushing is not fine enough, a proper percentage of the metals (remaining as they do locked up in their stony envelopes) is not secured. So that whether water be employed to concentrate them, mercury to entrap them, chlorine to attack them, or some chemical solution to extract them, there is a more or less considerable loss, which ought to be carefully studied, for the purpose of finding the

vanishing point (so far as it can be learnt) common to all these combined sources of trouble.

One may say then, that there is a certain average degree of fineness, to which it pays best to carry reduction in every process (varying with the ore, and the method of treatment). But before this point can be settled it is necessary to decide what is the best process for the purpose in view, *i.e.*, in getting the largest commercial return on a given capital outlay.

On these grounds, it is of paramount importance that the chemical and physical peculiarities of the ore and its associated gangue should be first studied, and carefully noted in connexion with the local surroundings of the works, in regard to the character and cost of labour, fuel, supplies, and abundance or absence of water-supply, as well as climate and other considerations, which must influence the choice.

To enter into minute details concerning the peculiarities of various ores, and the processes of treatment they might be subjected to, is far beyond the scope of this paper, but from a general standpoint, as affecting the present question, the materials (ore and country-rock and gangue) which the miner, the metallurgist, and the mechanic, have alike to deal with, may be divided according to their physical characteristics, in three chief classes :—

(*a*) Brittle and hard ; such as quartz, most pyritic ores, syenite, and coarsely crystalline metamorphic rocks.

(*b*) Tough ; such as native copper, gneiss, and most of the micro and crypto-crystalline metamorphic and intrusive volcanic rocks, which accompany ore-deposits.

(*c*) Soft or clayey ; these latter comprising earthy carbonates, surface ores, or clayey limestones, as well as some classes of stone, comparatively hard and brittle like schist but carrying clay-partings, and sandstone.

When the different physical peculiarities which attach to the first elementary mineral substance in the above list are considered, it is seen what wide room there may be for enquiry in this direction alone.

Silica which possesses no doubt a special interest to many engineers when it takes the form of gold-quartz, may possess under certain circumstances, what miners term a kindly appearance, tinged sometimes a light-grey, blue, or brownish-red colour, combined with a dark resinous look ; whilst elsewhere it may have a vitreous, wet, or opaque-white, hungry appearance, probably so-called by "cousin Jack," because it generally leaves him (if he attempts to work it) precisely in that very condition.

Still some of the best specimen-quartz is frequently milky, and white as a hound's tooth, whilst what is known as ribbon-quartz, a laminated variety of stone (separated by thin talc or chlorite-partings, or laminæ of graphitic shale or slate) is not unfrequently when met with, a good weight-carrier of metal. All these physical differences, due to peculiarities of hydration, association, and structure, will influence the way in which the stone breaks, and therefore affect its after treatment.

The materials dealt with by the mining engineer may, therefore, be divided according to their mineralogical characteristics and metallurgical behaviour, into a further number of classes, with different subdivisions:—

(*a*) Ores of the base metals, whether containing gold or silver or not, which have to be smelted or treated by a wet process, either with or without a previous (hand or mechanical) coarse or fine concentration, matting, or roasting, depending on circumstances which need not be enumerated.

(*b*) Ores of one or both of the precious metals, properly so-called, comprising:

1. Free-milling gold ores.
2. Free-milling silver ores.
3. Pyritic ores of gold and silver.
4. Combinations of 1, 2, and 3.
5. Exceptional ores.

These different kinds of ore may be subjected to various processes, amongst the most general of which, for dealing with free-milling* low-grade gold ores (belonging to class 1) are:—

1. Stamping the ore wet to fine size, and catching the free gold on copper plates (inside and outside the battery), with wells and mercury traps, used as accessories.
2. Wet stamping the ore coarse, and amalgamating on copper plates, separating the coarse-sands from the finer slimes, by screens or sizing-boxes (in some cases, preceding or following this separation by fine concentration); and regrinding the coarser sands thus saved, in Huntington or Chilian mills, or pans;† followed by more copper plates.

For dealing with free-milling comparatively high-grade ores of gold, belonging to class 1, in which there is an excessive loss, if the previously described systems of treatment are followed (owing to the fine

* A term implying that the greater part of the gold is free.

† Without mercury preparatory to re-amalgamation.

state of division of the metal, its alloyage with silver, or some other reason) there may be employed :—

1. Wet stamping, succeeded by direct pan-amalgamation on the ordinary plan (or else what is known as the Ross continuous system); or crushing dry (followed in exceptional cases by roasting) and then amalgamating in Chilian mills or pans. When there is a small percentage of rich sulphides present (especially if dealing with a mixed gold and silver ore) fine, *i.e.*, table-concentration, may precede or follow the pan-treatment with an appreciable saving.
2. In a few special cases, like the Mount Morgan mine, Queensland, ores of this class may no doubt be treated in bulk with advantage by drying, crushing with rolls, partially roasting, and working by chlorination in barrels. That the Mount Morgan ore may equally well be classed amongst the exceptional ores (belonging to class 5), becomes evident however, when the facts of the case are closely studied.

Mr. T. A. Rickard in a paper read before the American Institute of Mining Engineers, June, 1891, says :*—"The complete success of the treatment is largely due to the extreme friability of the ore, which renders its pulverization easy, whilst its porosity assists materially in the thorough chlorination of the gold." An enthusiastic writer has spoken of the ore as a sort of snow-drift, which melts in the chlorination vats of the company, into a golden sand, such as might be supposed to have been brought from the bed of the river Pactolus, instead of from the top of an Australian mountain. The extreme richness of the stone hitherto available for treatment in large quantities (the capacity of the works being 1,800 tons per week), the extremely minute state of subdivision of the gold, and the physical peculiarities of the ore above referred to, have all tended to make barrel-chlorination applied in this peculiar way a success, where the stamp-mill has been unsuccessful, and this also accounts in no small degree for successful crushing of the ore with rolls.

Mr. Rickard remarking on the cause of failure when the ore was stamped, says :—"The rock which the battery (25 stamps at Dee Creek) was called upon to crush, averaged 10 ounces per ton, but the contents of the tailings proved of much greater value than the amount of amalgam obtained. This led to a critical examination of the ore at the Sydney Mint." It was found that the bullion was of a fineness hitherto unknown in nature, assaying 99·7 per cent., occasionally 99·8 per cent., of pure

* *Trans. Am. Inst. Min. E.*, vol. xx., page 150.

gold, the rest being copper with a trace of iron. It is remarkable as being almost entirely free from silver. Dr. Lebius, of the Sydney Mint, considered as a result of numerous experiments, that the iron present was in the form of an oxide, which coated the gold, and so prevented its contact with mercury. In a footnote Mr. Rickard adds: "My own experience with the ores of Gilpin County, Colorado, leads me to believe that this is more frequently the obstacle to successful amalgamation than is usually supposed.

For dealing with free-milling silver ores (carrying less than 6 ounces per ton of white metal) belonging to class 2 as a rule one may employ :—

1. Wet stamping, followed by direct amalgamation in pans or barrels. The ores of this class are limited, only embracing decomposed surface-ores, carbonates, and occasional deposits of argentite, chlorides, chloro-bromides, and native silver.

For dealing with pyritic ores of gold and silver, belonging to class 3 which contain the larger proportion of their metallic contents in combination with sulphides. The following are the usual methods :—

(*a*) If the sulphides are massive.

1. Hand-picking and after treatment, as described below.

(*b*) If the sulphides (pyrites) are disseminated.

2. Wet stamping, combined with battery and plate-amalgamation, to extract the free gold followed by fine concentration of the pyrites, and treatment of the hand-sorted ore and concentrates by smelting for lead, which yields bullion that has to be refined by different processes; or by smelting for copper in reverberatory furnaces, to produce an enriched copper matte.* The choice of these methods depends on whether lead or copper is the predominating metal. More frequently, however, the gold is extracted from the concentrates by the ordinary Plattner vat-chlorination treatment, whilst in a few special cases an iron matte is produced, and sold to smelters, or crushed and chlorinated.
3. Wet stamping, followed by amalgamation and fine concentration, and grinding the concentrates raw, in pans or Chilian mills.
4. Dry stamping, followed by roasting, with or without salt, and amalgamation in pans or barrels, a process which is only adapted, however, as a rule, to rich silver ores, carrying a certain proportion of sulphides, but not enough to pay for concentration.

* The gold and silver in this matte, are subsequently extracted by a wet process from the copper-bottoms which contain the gold, after the silver has been extracted by the Ziervogel or Augustine processes.

5. The new cyanide-lixiviation-process, which is claimed to be applicable to the treatment of certain gold ores, much in the same way as the Russel process is to silver.
6. Ordinary lixiviation.

Dealing with ores belonging to class 4, which are amongst the most difficult to treat, because a process adapted to the extraction of gold may be very inefficient for the recovery of silver, and *vice versa:* when one metal is in small proportion as regards value, it is frequently sacrificed to the most profitable process commercially for the other. Many of the base silver ores of the United States contain 5 to 15 dwts. of gold, and from 20 to 50 ounces of silver per ton. For such ores dry stamping, roasting, and pan-amalgamation is mostly used, and under proper conditions will extract 90 per cent. or more of the silver, but only 40 to 60 per cent. of the gold. Lately, however, coarse-crushing, instantaneous roasting in Stetefeldt furnaces, and double-leaching by the new Russel process has come into prominence, and is said to be giving excellent results in certain instances, as regards economy and extraction of both metals.

The Russel process seems to be more particularly applicable to those ores which do not contain enough lead or copper for smelting, are poor both in gold and silver, and which contain large amounts of sulphur, arsenic, and antimony, since roasting with salt would convert the base metals as well as the silver into chlorides, and would give a very base bullion if it were amalgamated. There is, however, a remedy for this, which Mr McDermott points out*: "When the silver is in combination with sulphides, antimonides, arsenides, and tellurides of the baser metals, the pan process becomes inefficient and expensive. By contact with iron surfaces, heat, and the addition of some chemicals—chiefly salt and sulphate of copper—a partial decomposition of the complex minerals is effected, and some of the silver amalgamated, but the wear of iron, loss of mercury, cost of power and chemicals, and the production of base bullion together, go far to neutralize the gain made in the recovery of the silver. In such a case a great benefit is derived by combining concentration with amalgamation. The light, flocculent chlorides and sulphides of silver can be amalgamated to a high percentage, while concentration is almost useless to deal with them in the form in which they exist in a free or decomposed ore.

On the other hand, concentration can be made very effective on the undecomposed complex minerals, for which amalgamation is ill-adapted.

* *Gold Amalgamation*, by Walter McDermott and P. W. Duffield, page 88.

There are two methods of combining the processes according to their order, concentrating either before or after the pan-treatment. In regard to the relative advantages of the two, concentration before amalgamation is the natural method, because it relieves the pans of the baser minerals, which are a disadvantage in the amalgamation, and the subsequent concentration of which is made more difficult by the grinding or attrition of the minerals in the pans.

The only argument against the universal adoption of this order, rests on the disadvantage of sometimes having native metals, and some chlorides and sulphides entering the concentrations, instead of appearing at once as bullion, which they otherwise would do, and also, that very perfect settling of the slimes from the concentration tails is necessary, to prevent loss of the flaky silver chlorides and sulphides. This process is in use at the Standard mill, California.

When the free metal is gold, the first-named disadvantage can be overcome by using copper plates before the concentrators, and this process was adopted by the Montana Company with great success, after first trying pans before the concentrators.

Comparing the combined process with dry crushing and roasting, the advantages of the former consist in a reduction of the cost of working fully one-half of that of dry crushing, a crushing capacity of double for the same number of stamps, a decreased cost of erection, and a higher saving of gold present. Against these advantages may be placed simply the increased percentage of silver saved by the dry process under certain circumstances.

When combined gold-and-silver ores carry over 10 per cent. of base metals it usually happens that the silver does not exist equally in the minerals present, but is concentrated in one of them as a rich, brittle ore. It follows from this, that concentration in some instances is very ineffective, because while 90 per cent. of the heavy base metals may be saved in the concentrates, the loss of the fine silver-bearing mineral in the slimes may be fully 50 per cent. of the assay-value of the crude ore. In some cases the clean concentrated, heavy minerals do not assay more than the original ore from which they were separated.

This is a very strong argument against the injudicious application of fine concentration to all cases, for, as Mr. McDermott says, there are ores of both gold and silver in which the sulphides are so rich, that even a very small loss by weight involves a very large loss by assay of the precious metals.

Such minerals as tellurides of gold and silver, ruby and brittle silver

ore, come under this head. In concentrating some of the tellurium ores of Colorado, some of the very finest slime-material, overflowing from the concentration-tanks of the Frue vanners, assayed as high as £5,000 per ton in gold and silver.

Where there is danger of loss, as described, it is advisable to introduce a double concentration, re-treating the tails on vanners.

For dealing with exceptional ores belonging to class 5: if their value lies mostly in tellurides of gold and silver, with little or no sulphides present, the most successful method yet found is by selection of the rich ore; and concentration (as above described), by double treatment of the low-grade ores. Both rich ore, and concentrates, are then so valuable, that smelting on the spot or shipment to smelters is advisable.

Another class of exceptional ores are those, in which the gold is not free, so far as tests indicate, and yet there are no sulphides present.

These ores are generally oxidized in character, changed from their original form, and contain chloride of silver in addition to the gold. As some of these ores, when leached with hyposulphite of soda, yield a large portion of the gold as well as the silver, it would seem to indicate that the gold exists as a chloride, combined with the silver, either as a double insoluble chloride, or mechanically protected by the latter from being leached out by the natural drainage-water of the mine. Such ores are uncommon, but have been worked to some extent by a raw-leaching and concentration for the fine carbonate of lead, and its combined silver which is also present.

There may be, other explanations, however, to account for the peculiar character of the ores just alluded to. On page 6 of *Gold Amalgamation and Treatment*, the losses of gold in milling are referred to five principal causes:—

1. Loss of free gold, quicksilver, or amalgam, due to carelessness or inexperienced amalgamation.
2. Free gold and gold-bearing sulphides, attached to or embedded in particles of rock.
3. Gold contained in base-metal sulphides, broadly termed sulphides.
4. Gold lost in fine slimes, and sometimes in solution.
5. A condition of gold in which it is not susceptible of copper-plate amalgamation.

It is with the last of these that we now have to deal.

Quite recently * Mr. Richard Pearce, of Argo, Colorado, has given

* "The Association of Gold Ores with other Metals in the West." *Trans. Am. Inst. Min. E.*, vol. xviii., page 447.

some most interesting facts with regard to the association of gold with other metals in Colorado, which bear on amalgamation and concentration. His investigations go to prove that gold and silver exist in some of these ores found below the water-line, alloyed with bismuth, tellurium, and copper, and perhaps arsenic, where the presence of such combinations has hitherto been unsuspected, and a coarse-grained almost pure pyrites containing silver and a small value of gold, from the Louisville mine of Leadville, has been proved to contain metallic tellurium. When thus combined the gold would appear to be more or less refractory to the ordinary milling treatment, and this may account for some of the troubles set down to so-called rustiness.

A natural crystalline compound of gold and bismuth, has lately been discovered in Australia, it is said, and native alloys of these metals are known under the names of maldonite and bismuthaurite. Hessite, a natural combination of tellurium and silver (Ag_2Te) has been found in some of the Red Cliff veins, and a telluride of bismuth, probably tetradymite was found by Mr. Pearce in ore discovered at Ouray. He contends that, in certain cases, bismuth performs, a by-no-means small part, in influencing the general deposition of gold, and also in causing the refractory behaviour of this metal under metallurgical treatment.

Mr. Pearce adds: "The occurrence of gold in this district seems to be intimately associated with that of eruptive porphyries, and the impregnation of veins with gold-bearing minerals is apparently always accompanied with silica. Evidence of intense thermal action* is one of the chief characteristics of gold deposits. Examples of these processes may be found in Gilpin, and also in Boulder County. In the latter district known as the 'telluride Belt,' there are indisputable evidence of vein-impregnation by the circulation of siliceous waters through the joints of the porphyry; in this case tellurium is the chief mineralizing agent of the gold. Perhaps the most striking example of the deposition of gold, through the agency of thermal waters, may be seen in the celebrated Bassick mine, which has so often been described. This deposit exhibits all the characteristics usually accompanying geyser action. The disintegration of the porphyry, with a partial replacement of the felspar by silica, is here clearly shown. The gold was combined almost in its entirety

* The idea Mr. Pearce intends to convey (which is the writer's view of the matter) is that the filling of veins with mineral, is rather due in most cases to the leaching of the adjacent rock-masses, followed by a gradual replacement of the original constituents of the lode with ore, along certain horizons or belts, than to the action of thermal waters, building up masses of mineral in an open fissure, though both solfatara action, as Mr. Pearce remarks, at the Bassick mine, and sublimation in exceptional instances, may have tended to such a result.

with tellurium, and rich tellurides of gold and silver were found sprinkled through the mass of material. In some remarkable specimens these tellurides formed a distinct coating on the surface of smooth boulders which had become rounded by attrition from the action of steam."

Dr. T. Egleston, in an elaborate paper on the causes of rustiness and some of the losses in working gold ores,* points out that the term rusty is used by miners to indicate a condition in which the metal is supposed to be coated superficially or alloyed with some substance, which prevents contact with mercury, and consequently precludes the possibility of amalgamation.

Gold in this state, it appears, is frequently covered with a brownish coating, which has a much redder colour than ordinary gold, and is irregularly distributed over its surface: where the least abrasion has occurred the metal underneath show the true gold colour. Fine particles of gold are sometimes visible with the microscope in the detached coating, and it often cracks off from pieces of gold, leaving them bright. Very often this film is composed entirely of silica, deposited on and beside the gold. It is sometimes opaque, and again quite transparent, so that the gold can be seen with the microscope disseminated through it, just as cinnabar-crystals are seen in the red chalcedony of the district around Knoxville, California. There may be, for example, many artificial causes which produce this rustiness of gold, the covering of the surface with particles of some foreign substance, or its alloyage with other metals, as already explained.

As Mr. McDermott remarks, concentration and pan-amalgamation would probably be effective on the former class of ores, though for reasons that will presently appear the writer would like to add, or concentration preceded by crushing in centrifugal roller-mills, if other circumstances admit of it. Dr. T. Egleston's view is, that the action of rubbing, which occurs in any machine like the arrastra, is much more likely than the stamp-mill to pulverize the fine pyrites, break up any coating that may be around the particles of gold, rub off the superficial deposit, and thus bring the gold into contact with the mercury and make it amalgamate. He states, as a remarkable fact, that in the early days, Mexicans with the arrastra, got 50 to 60 dollars a day, where stamp-mills, working the same rock, only obtained 15 to 20 dollars, and instances can be cited where, with the best modern machinery, only 20 to 30 dollars can be got out of rock which yields 700 to 800 dollars by fire assay.

* *Trans. Am. Inst. Min. E.*, vol. ix., page 646.

Of course this might be due partly to the first cause of loss cited, viz., carelessness or inexperience on the part of the workmen running the mills, but this is on the whole unlikely. Amongst other causes in the treatment and composition of ores which render gold un-amalgamable in the mill, and which have to be considered and guarded against, Dr. Egleston mentions :—

1. Gold which has been hammered, so as to increase its density and close its pores, amalgamates very slowly, a condition, however, which can scarcely be taken into account in the working of a stamp-battery, unless the gold is very coarse.
2. The presence of sulphuretted hydrogen in the water, likely to be induced by the presence of soluble sulphides.
3. Gold which has been exposed to the vapour of sulphur, which no doubt accounts for the behaviour of certain ores when submitted to amalgamation after roasting, and the same is doubtless true of arsenic in some instances, judging from Mr. Pearce's conclusions.
4. Certain alloys of gold, like the phosphide, though others like the arsenide and antimonide (produced artificially) are apparently more tractable. To these alloys must now be added the bismuthide, telluride, and cupride.
5. Greasy substances, like powdered natural hydrated silicates of magnesia and alumina, if present in the ore, froth and coat the gold with a slime, preventing the action of the mercury, and lubricants such as oil, getting into the battery box, are fatal to gold amalgamation.

The late Mr. E. N. Riotte once mentioned in the writer's presence a case within his experience, which occurred at a gold mine in one of the Southern States, where they had to abandon the use of dynamite on this account; as owing to the inexperience of the negroes in its use, portions of unexploded cartridges were left in the stone, and affected the battery yield. The result, if the nitro-glycerine had leached out, and collected in a corner of the box, would doubtless have been a revelation to them.

6. Unusually soft water, absorbing air or carbonic acid gas, and so tending to form ferric sulphate (when pyrites is present), which is a most active agent in staining the plates.

Glancing back at the processes that have been referred to, it will be seen, that except in dealing with free-milling silver ores and a few special cases, there is considerable latitude given in the choice of a process, not to mention its details.

Its proper selection will depend on considerations of relative cost, as compared with saving or loss of metal, or other valuable products, that the ore contains; and nothing influences both of these in the same way, or either of them more, than the degree to which the crushing of the stone is carried, which is the author's next proposition.

The finer the crushing, and the more the ore is handled or re-handled, by repeating the operation, the greater of course will be the expense; while the more it is handled and the finer it is crushed beyond a point that is absolutely necessary, the greater also will be the loss. To reduce this latter to a minimum, a certain cost must naturally be incurred, though if it go a little too far, the cost is increased and the loss as well.

This happens under various circumstances, viz. :—

A. In coarse concentration of the baser metals and of gold and silver ores, or a combination of both (applicable to the latter classes of ore, when the sulphides exceed say 10 to 20 per cent. by weight) to obtain the mineral with a minimum loss from comminution, the ore must be broken up, only just enough to unlock all the mineral it may contain, down to a certain size, and (theoretically) the mineral and barren rock with it should be separated at once. This will leave, however, a certain amount of yet more finely divided mineral in the residues, which require to be re-crushed and re-treated at so much additional cost, to get out the metal that remains in them, less of course a certain percentage of loss involved in this re-treatment.

When coarse combined with fine-concentration is necessary, it may then be considered how far it will pay, to carry theory into practice, and when labour is cheap, the life of the mine assured for a long period, and the ore of considerable value, the German practice may be adopted, of classifying carefully, reducing the ore gradually, and concentrating close; but when labour is dear, the life of the mine uncertain, and the chief object is to secure the largest possible profit in the shortest possible time at a minimum capital outlay, irrespective of a certain unavoidable loss in unrecovered mineral, the Anglo-American method must be adopted of sacrificing saving of mineral, to saving of cost and time, and crush the ore in quantity, to a much greater average degree of fineness, without such careful regard to sizing it.

The latter practice is the one followed in most American and colonial mills, employing fine concentration for gold and silver ores (when the sulphides in the ore do not run over, say, 10 per cent.). To compare

these divergent systems roughly as to cost, two different cases, one of a European zinc-and-silver lead-mine, the other an American lead-mine may be taken.

The works in Europe are those of the New Pierrefitte Co., which the writer was commissioned to inspect last summer, and of which he is permitted to give the following particulars :—

They are situated in the Pyrenees in France, and treat an ore containing galena, blende, magnetic iron, and a small amount of copper pyrites.

On an output of 21,449 tons of crude ore last year (averaging 9·33 per cent of silver-lead, and 11 per cent. of zinc) the cost of dressing, covering wages, supplies (including coal for three months of the year, water-power being used for the remainder), renewals of machinery, etc., may be taken at about 3s. 10d. per crude ton, as given below :—

Continental Works. Per Crude Ton. Francs.		American Works. Per Ton. Cents.	
Labour	2·35	Labour	13·4
Supplies and repairs ...	2·43	Supplies	3·5
		Repairs	10·0
		Coal	9·5
Total ...	4·78 = 3s. 10d.	Total ...	36·4 = 1s. 6¼d.

The American case is that of the new dressing works of the St. Joseph Lead Co., at Bonne Terre, Missouri.* The ore is non-argentiferous galena, associated with some iron pyrites, carrying traces of nickel and cobalt. On an output of 224,203 tons of crude ore in 1886-1887, averaging 5·65 per cent. of galena, the cost of dressing for the fiscal year, ending May 1st, covering labour, supplies, repairs, and coal, was (1s. 6¼d.) 36·4 cents, divided as above shown.

It would take too long to explain the differences of treatment in the two mills, suffice it to say, that the system outlined in both, is to crush with rock-breakers and rolls (preceded at Pierrefitte by spalling and cobbing), and then to jig and wash the ore on tables and buddles. The difference mainly is, that in the one case they give attention to sizing, and in the other they do not, though each mill in detail possesses certain peculiarities of its own. Neither can the relative number of workmen, cost of wages, supplies, etc., in detail be compared, as these data in the cost sheet are wanting, in the otherwise excellent paper quoted, from which the particulars of the Missouri mill are taken. In the French works, wages range from 1 fr. (10d.) to 2 fr. (1s. 8d.) for boys, and 2·75 fr. (2s. 2½d.) to 3 fr. (2s. 6d.) per day for the larger number of adult

**Trans. Am. Inst. Min. E.*, vol. xvii., page 659, H. S. Munroe.

mill hands, and it is noteworthy that in both mills repairs and supplies (excluding coal from the American case) bear nearly the same ratio to one another.

It would appear at first sight, that all the advantages lay with the American mill, and no doubt it does in America, where labour is dear, with the base ores that have to be dealt with in quantity, where the works are situated. The case, however, would be entirely different if the American mill were transplanted to the Pyrenees, and set to work the silver ores of that locality, as the loss in the tailings at Bonne Terre is about 2·13 per cent., or 27·4 per cent. of the total amount of lead in the ore. These large losses are due to included mineral in the coarse-sands, and to very finely divided mineral in the very fine slimes, as a large part of the ore requires very fine crushing, owing to the exceedingly minute state of division of a large part of the mineral.

In the Pierrefitte works such losses with argentiferous galena could not be afforded, and though the writer is not in a position to state what they are exactly, they are far less than in Missouri. Moreover, the cost in the Pyrenees, with a larger output from the mine, could be materially reduced, as the works are not run up to their full capacity, and if their general design could be remodelled on the lines of more modern Continental mill-construction, the expense of treatment (if carried out on a similar scale as regards quantity) might, be made to compare more favourably with the price in America without sacrificing mineral, in view of the lower price of labour.

To again quote Prof. Munroe, the losses in the Lake Superior copper mills range, it is said, from 28·5 to 31 per cent., treating mineral much more easily saved than the Bonne Terre galena, whilst the cost of dressing copper at the Atlantic mill in 1885, using steam stamps, is stated to have reached 30·36 cents (1s. 3½d.), divided between labour, 5¾d.; fuel, 7¼d. ; supplies, 2½d. These costs were reduced, however, at the Atlantic mine in 1887 to 27·5 cents or 1s. 1¾d. per ton of copper ore treated. Incidentally, Prof. Munroe states it is cheaper to crush ore* with rolls, arranged as they are in the Bonne Terre works, than the Lake Superior amygdaloid with steam stamps, but this is partly due to the friability of the Missouri ore ; what is far more remarkable, however, is that the rolls produce (according to the Professor's statement) quite as large a proportion of slimes as the steam stamps.

Further, it should be noticed that the above instances of the cost of concentration in America are to be looked upon as a good deal below the

* Limestone.

average (at any rate of smaller mills), and as a general rule, coarse concentration, where rolls are employed for crushing, does not cost far short of 75 cents or 3s. 1½d. per ton, where labour rates are dear.*

The average cost of dressing different classes of ore in France is given by Dr. Foster and Mr. Galloway† (from a statement of Mr. Huet) as:—

	Per Ton. s.	d.
Iron ore	0	2
Coarse-grained galena	5	10
Fine-grained galena	7	10
Manganese ore	7	10
Copper pyrites, or grey copper ore with iron pyrites	9	9½
Coarse galena and blende	9	10
Fine galena and blende	11	10
Copper pyrites, or grey copper ore with galena ...	13	10
Copper ore, galena and blende	20	4

Though these figures may have been typical and true of French practice at the time they were compiled, the dressing-costs at Pierrefitte would indicate that they are much above the average at the present time; and, as the translators remark, there is always great uncertainty in estimating such working costs in a general way, for there is always uncertainty with reference to the labour required for shifting the stuff about, and principally with reference to the loss in dressing, to which might be added the attention required by different classes of machines when differently grouped.

If the American losses in dressing be compared with those in Germany it will be seen that the advantage rests with the Continental system.

Oberbergrath O. Bilharz, in a paper entitled "Die Neue Central Aufbereitungs-Werkstätte der Grube Himmelfahrt, bei Freiberg i.S.," puts the losses in the tailings of these works, at only 0·01 per cent. silver, *nil* per cent. lead, 10 per cent. sulphur, and 9 per cent. zinc. While the settlings in the final catch-pits‡ carry only 0·01 per cent. silver, 2 per cent. lead, 8 per cent. sulphur, and 6 per cent. zinc.

Quoting the pamphlet referred to, the transcript of an article from *Industries* (in the *New York Mining Journal* of February 20th, 1892) and the writer's personal knowledge, he would like to refer to the Freiberg works more particularly, because he thinks they are illustrative of American principles, so to speak, engrafted upon former German practice,

* In exceptional cases the price may run up to 12s. 6d.

† *Lectures on Mining*, by J. Callon, vol. iii., page 134. Translated by C. Le Neve Foster, D.Sc., and W. Galloway.

‡ Only a fractional portion of the crude ore, which is re-treated.

presenting a model of economy in costs, as well as economy in saving of mineral, to an extent which has never been achieved before, an advance which is undoubtedly in the right direction.

The ores which are obtained from the various shafts of the Himmelfahrt mine consist of argentiferous galena and zinc blende, with iron, copper, and arsenical pyrites, while the gangue is partly quartzose, partly sparry, mixed with the country (gneiss).

On account of the variety of the ores of the Freiberg district, the machinery of the dressing-floors is duplicated, so that ore from other mines can be dressed apart in the two sections, into which the mill is divided.

The annual production of the Himmelfahrt mines alone is 45,000 crude tons, of which four-fifths is concentrating ore, coming from what is known as the pyritic-lead formation (kiesige Blei-Formation) the remainder being rich silver ore and high-grade hand-picked galena ore, found mostly in cross-courses, carrying a gangue of brown spar and heavy spar.

These latter ores, representing one-fifth of the total production, are separated dry (hand-sorted and dry-stamped) and sent direct to the smelting-works.

The dressing-floors* are designed to handle 150 tons of pyritic ore per shift of ten hours, each separate section dealing with 75 tons, with an average total consumption of 35 cubic feet of water per minute, the machinery being driven by steam-power.

The ore averages 0·15 to 0·20 per cent. silver; and iron pyrites and galena are the predominating minerals. The zinc blende is black, and contains 33 per cent. of iron. The building is arranged in storeys, one above the other, and contains rock-breakers, screens, hand-picking tables, several sets of rolls, wet gravitation-stamps, a variety of revolving screens and jigs, hydraulic classifiers, Stein vanning-frames, and other machinery (fully described in the papers referred to) which present many novel features of detail.

The horizontal compound-condensing engine which runs the whole crushing, dressing, and electric-lighting plant, indicates 105 horse-power, and the works employ 44 workmen, with 3 overseers, 1 engine-driver, 1 stoker, and 5 fitters, in all 54 men.

The cost of dressing one ton of ore amounts to 10d.

* The machinery was built and erected by Mr. C. Luhrig, of the firm of Messrs. Luhrig, of Dresden, in accordance with the plans of Mr. O. Bilharz, who designed the installation.

There are only, five points which could be charged against the plant taken as a whole :—

1. That the design involves a considerable capital outlay (an objection, which it will be shown later on, is entirely swept away, when it conduces to and results in economy of treatment, provided that the mine warrants it, and that the working capital of a company can stand the necessary call without stinting the development of the mine.)*
2. That it may be doubted whether efficient roller-mills of the Schranz type, might not advantageously be substituted for the stamps.
3. That, large as may be the reduction in labour as compared with many other Continental plants of the same capacity, it might be still further lessened, given a better site than the one at Freiberg, which possesses but a moderate fall.†
4. That owing to the nature of the machinery, several skilled machinists, as well as facilities for making repairs on the spot, are very essential for running it properly.
5. That if anything goes wrong with any of its individual parts, the whole plant appears to be brought to a standstill. The writer was reminded of this by the practical illustration of a belt slipping off a pulley whilst he was going through the works two years ago. It is a matter, however, which is capable of remedy.

On the whole the advantages far outweigh any objections that can be urged, and the structural details of the plant, and erection of the machinery exhibit first-class skilled workmanship.

The stock required to be kept on hand in concentration-works consists of duplicate parts of the machines, in more or less number, (depending on the distance from a source of supply and the average life of the different pieces in use).

The proposition stated on page 109 holds good.

B. In the fine concentration of sulphides (which generally accompany the precious metals) on belt or other automatic concentrators, which are so frequently used‡ as an adjunct to the stamp-battery.

* As these works are a Government concern, this last consideration may in the present case be dismissed.

† Owing no doubt to its being the only site available on account of other reasons.

‡ Before adding concentrators of this kind to a mill, it is of course necessary to ascertain that the clean concentrates will assay enough to leave a profit on further local treatment, or shipment, to extract the gold, silver, or other metals they may contain.

The size to which the stone is reduced under such circumstances influences very largely the chance of loss of mineral and gold, whether included in the coarser part of the tailings, or liberated as fine slime.

The cost of the actual mechanical fine concentration is not a matter of extremely serious consideration in itself, since assuming 18 tons of concentrates saved *per diem* in this way, crushing an ore containing $2\frac{1}{2}$ per cent. of pyrites, the cost of treatment would not exceed 2d. or 3d. per ton, and dealing with much smaller quantities, 6d.; and further it will not vary much whether the ore is stamped coarse or fine, unless it involves a double concentration (that has been alluded to) dealing with rich ores liable to slime. But in order to concentrate in this way the ore must be stamped wet, and what applies to crushing fine with rolls, is also true of stamps. The finer the crushing the more it must cost in time, representing extra labour and a wear-and-tear, and also fuel, etc.

C. In stamping previous to grinding in pans, if carried too far, there will again be loss in the battery-tails, whilst if not carried far enough, it will give the pans unnecessary work to perform, which would be much more cheaply done in the battery.

D. When the concentrates are treated (obtained by coarse or fine concentration or both combined) by such processes as smelting, chlorination, and lixiviation, it will be found that, if it involves a preliminary roasting (as these processes generally do), if the ore is crushed very fine, and it carries much lead for instance, it entails a more careful regulation of the temperature, and unless a suitable type of furnace is used, with extensive dust chambers attached, great losses both mechanically and by volatilization may be incurred. On the other hand, if the ore is not crushed fine enough, and it is subjected to a chloridizing roasting with salt, the chlorination will not be as high as it should be, and the subsequent treatment will be proportionately affected by it.

E. If the ore is to be lixiviated or chlorinated, and the stone is crushed too fine, and at the same time of too uniform a grade, great loss of time and imperfect filtration and leaching will result, more especially if it is at all of a clayey nature.

The choice of a process, whether for treating crude ore or concentrates, must be governed by the relative commercial profits it promises, as compared with other methods of treatment. But before attempting to touch

this question, it will be advisable to state certain general facts (under separate headings), giving an outline of several of the processes, which are later on compared, under the head of rival methods of ore-treatment, in order to simplify the subject.

Roasting and Chlorination.

This is, ordinarily the best process to employ, for the treatment of the high-grade pyritic concentrates of stamp-mills, and hand-picked sulphide ores, provided a sufficient quantity can be commanded to run the works full-time or nearly so; intermittent work being bad for plant as well as for men. The ore should be kept moist after concentration, till it goes to the drying-floors, as otherwise lumps will form which will not roast properly unless re-crushed.

The ore to be roasted is generally worked in reverberatory furnaces to expel the sulphur, arsenic, and other volatile compounds till it is dead sweet or as nearly so as possible. Then a small quantity of common salt is added, and the silver is chloridized, the sulphur is all driven off, and the ferrous and cuprous sulphates are oxidized.

The ore having been damped to about 6 per cent. is gassed with chlorine, a trichloride of gold being formed, which is leached out and precipitated with a solution of sulphate of iron. This gold is collected on filters, thoroughly washed, dried, and melted, and should average from 998 to 999½ fine.

When silver is present in sufficient quantity to justify the extra treatment, it can be recovered by re-leaching the residues with hyposulphite of soda or lime to dissolve the chloride of silver, which is precipitated from the lixivium by the addition of a solution of polysulphide of sodium or calcium, and the sulphide of silver resulting (collected on the filters) is washed, dried, and reduced to the metallic state.

There is, however, one point to be guarded against in adopting this latter mode of treatment in dealing with ores, in which the percentage of gold is high, and about equal to the silver.

Kustel,* remarks:—"If such an ore should be subjected to chloridizing roasting, then impregnated with chlorine gas, leached with water for the purpose of extracting the gold, and finally leached with hyposulphite of lime for precipitating the silver, it would in this case, although a high percentage of silver might be extracted, result in a yield of gold that would hardly amount to 50 per cent., more or less.

* *Roasting of Gold and Silver Ores*, by G. Kustel, second edition, page 123.

The reason is not easily explained. The gold may be influenced somehow by the base-metal chlorides during the roasting, which prevents the gold being attacked by the chlorine gas."

On the other hand, if the base-metal chlorides and the chlorides of silver are extracted previous to the impregnation with chlorine, both metals, silver and gold, can be got out very close by a process invented and patented by O. Hofmann, which Kustel fully describes in detail.

The gold and silver-bearing sulphides of the Colorado No. 2, G. and S. M. Co., at Monitor, Alpine County, California (as an instance) were successfully treated by this method.

Should the sulphides contain any lead, it is advisable to conduct the liquor from the leaching-vats to settling or storage-tanks, and about 40 lbs. of sulphuric acid (66 degs. B) is added. By this addition the lead is precipitated as sulphate, and the liquor, being freed from lead, can yield no plumbic sulphate with the gold, as it would otherwise do, when precipitated with sulphate of iron; hence a cleaner bullion.

In some chlorination works,* as the sulphate of lead obtained from the base tanks, always contains some gold, it is collected and sold, and a considerable sum is then realized for both lead and gold. The wooden tanks are protected from the action of the acids, by a coating of paraffin paint; and the covers of the chlorination-vats are luted on with a mixture of tailings, bran and water.

There are certain pyritic ores, however, to which chlorination is unsuited. These include amongst others :—

1. Ores of low grade, which run below, say £3 to £4 value per ton.
2. Those in which the gold is very coarse.
3. Those in which there is any mineral present, or other metal except silver and gold, in the gangue, liable to be attacked and rendered soluble by the chlorine, whilst gold must be either in a metallic state, or in a combination which can be destroyed by roasting without loss.

The impurities most to be avoided are sulphur, antimony, and arsenic, since soluble salts of the base-metals may precipitate the gold in the leaching-vat; hence the necessity of what is termed a dead sweet roast before-mentioned. Lead, lime, and magnesia are also deleterious, as they are attacked by the chlorine, and waste a great deal of it, besides introducing difficulties in the after-precipitation.

Mr. Nelson E. Ferry, M.E., in the *New York Engineering and Mining Journal* of November 28th, 1885, recommended the addition of molasses

* *Eighth Report of the California State Mineralogist*, page 47.

to the leach when lime was present. Mr. Ferry says: Dissolve one gallon of molasses in 30 or 40 gallons of water and keep for use. The quantity to use in each case must be determined by a laboratory test. If calcium sulphate comes down, either the molasses is in insufficient quantity, or it has not been thoroughly mixed. Examine by transmitted light. Avoid large excess of ferrous sulphate. If the gold comes down at first in a flocculent state, that does not matter, it soon assumes the usual form. The best results are got when the liquid is made slightly acid.

Mr. A. H. Aaron* recommends the use of precipitated copper sulphide as a precipitant,† stirring it into the gold solution, or better, allowing the gold solution to flow through a series of small filters containing the sulphides. He states as a reason that, unless there is copper present to precipitate it, there is often a considerable loss from gold in suspension, due to imperfect settling.

The presence of galena, necessitates a good roasting with a strong finishing heat, as far as its fusibility will allow. This involves the use of a long hearth to raise the temperature gradually, as the charge progresses from one end of the furnace to the other. The roasted ore must be examined to see that no galena is left undecomposed.

Any soluble iron chloride or other soluble metallic salt formed, will react on the iron oxide and precipitate the gold, when dissolved, leaving it in the tails of the ore-tanks after lixiviation. The addition of a little salt towards the close of the roasting, after the ore has been carefully oxidized, tends to counteract the effect of the lime and magnesia so often present in gold and silver-bearing rocks.

Concentrates frequently contain iron as well as copper pyrites, galena,

* *Notes on the Hydro-metallurgy of Gold.*

† Mr. Claude Vautin, whilst admitting that precipitated copper sulphide (CuS) is a good precipitant in the laboratory, points out that in consequence of its physical condition, and the facility with which it is oxidized to $CuSO_4$, its application in practice is not to be recommended. He recommends fused sub-sulphide of copper (Cu_2S) as a perfect and rapid means of recovering the gold. To facilitate the filtration the Cu_2S is crushed to pass a 60 mesh sieve, and the portion which remains on a 100 mesh sieve only is used; this gives the desiderata of a dry, hard, and granular filter-bed, three points of importance. Whilst charcoal is not free from objections, on account of the large quantity necessary to ensure a complete decomposition of the solution, and the somewhat tedious and troublesome operation of "burning-off" the excess of carbon before smelting the residues, Mr. Vautin claims for this method the advantage that it is not necessary to expel by heat or otherwise any excess of free chlorine from the solution before passing it through the reagent; nor does the presence of any free hydrochloric acid in any way interfere with the reaction. Any copper, calcium, zinc, etc., also passes through, presenting advantages over the use of hydrogen sulphide or ferrous sulphate.

and arsenical pyrites, combinations which do not prevent the application of the process, unless the base-metals, more particularly lead (but always excepting iron), are present in large proportions. Losses are likely to occur through the gold being volatilized or affected in roasting, which is more particularly to be guarded against if copper is present, and the ore be roasted with common salt. Much has been written on this subject, and in fact, enormous losses may occur in the chloridizing roasting of gold ores through volatilization of the gold.*

Mr. Kustel, page 57, records the loss of 20 per cent. of the gold contents in the oxidizing-roasting of certain tellurides of gold and silver, and states it is not a mechanical loss, but is due to volatilization. Though with most ores no loss of gold is suffered in this way during oxidizing-roasting either with iron or arsenical pyrites, loss may occur if the operation be carried on so rapidly that fine particles are carried off by the draught.

A loss of silver in oxidizing-roasting is unavoidable. Plattner† concludes that the percentage increases with the temperature of roasting, and with the looseness or porosity of the roasting charge, that is with the facility with which the air can come into contact with the silver, and the freedom of the silver from combination with other substances. The loss increases also with the time of roasting. He concludes that silver is volatilized as oxide, which decomposes at a lower temperature into silver and oxygen, but Dr. Percy throws doubt on this theory.

In roasting, it is easy to incur an enormous loss of gold by inattention to what may appear insignificant trifles, showing how necessary in such matters is a systematic weighing and sampling, as well as assaying of the ores and products, in order to know at once when such losses are taking place, so as to be able to check them in time. Haphazard and occasional sampling and assaying, are worse than useless; they lead to great losses of valuable capital, frequently to the total abandonment of good properties, and worse than all, to a false sense of self-satisfaction, which discourages investigation and improvement by denying the necessity thereof.

If the gold in the ore is of low fineness or combined with silver, a large part of it may be lost in roasting from a different cause, as the sub-chloride will surround the particles of gold, and prevent any further action of the chlorine upon it, in which case the double process, previously alluded to, must be employed.

The quantity of common salt used in chloridizing-roasting, and the time when it can best be added, are of much importance to the final result;

* "Losses in Roasting Gold Ores and the Volatility of Gold." *Trans. Am. Inst. Min. E.*, vol. xvii., page 3.

† *Metallurgische Rostprozesse*, Freib, 1856.

and they have to be determined by trial, with different ores. The chlorine must be purified from everything that would be likely to cause a reaction between it and the other constituents of the ore.

Other important factors in the general success of the process are :—

1. Close and clean concentration.
2. Amalgamation of any coarse gold, preceding or following the chlorination.
3. Lixiviation of the silver before or after gassing.
4. Absence of any organic matter in the charge, or wash-water.
5. That the ore is crushed properly. A series of experiments should be made to ascertain how coarse the ore should be to give the best results with reference to economy, large capacity, and best extraction. The pulp for the best leaching must be in a granular condition, and carry as small a percentage of dust and slimes as possible. Mr. John E. Rothwell (*New York Mining Journal* of February 7th, 1891) points out that for this purpose rolls, properly managed, are best adapted for this purpose. The chief point is to make the reduction in the size of the particles passed through them gradual. The ore should come to the coarse rolls not larger than $\frac{3}{4}$ inch, and these rolls should crush to about $\frac{3}{8}$ inch. The middle rolls are set about $\frac{3}{16}$ inch or less apart, and the fine rolls about as far apart as the size to which the ore must be crushed. If only two sets of rolls are used, the coarse are set a little closer, and the fine remain the same. The springs should be tight enough in tension, not to give with the hardest ore passing through them, but lax enough to allow a piece of steel or iron to pass through without throwing off the belts. The periphery speed of the rolls should be the same or a little faster than the falling speed of the ore, and the ore should be fed in an even sheet across the surface of the roll. This will keep the surfaces true and produce a granular pulp but carrying a small percentage of dust. A still more gradual reduction can be made by making the rolls of larger diameter and narrower, which will give them also proportionately a greater capacity. Rolls of 39·5 inches diameter and 12 to 15 inches face have been used by Mr. Rothwell with good results.

Dr. T. Egleston writing on the formation of gold nuggets and placer deposits says :—"I have known of gold (Grass Valley, California) being

thrown down on the filter of a Plattner vat by the organic matter contained in the very impure wash-water used for the solution of the gold, rendered soluble by the action of the chlorine. The filter was full of metallic gold, and there was no means of ascertaining how much had been lost. Several ounces of a brown deposit were taken from it, almost pure gold."

An excess of chlorine must be present in the generator, and the ore must be washed and filtered thoroughly, otherwise the tub tails will assay 4 to 5 dollars instead of 75 cents.*

Lixiviation.

The MacArthur-Forrest Process for Gold Ores.

This process claims† to extract the gold from pyritic ores, without the necessity of roasting; and in certain cases it may have a considerable field of usefulness. When, for instance, the gold is in an extremely fine state of division, and the ore contains silver as well as gold, the fairly high percentage extracted of each metal may render this method a very desirable one.

For plain gold ores, in which the gold is in fine particles, the barrel-chlorination process seems, however, to give a higher extraction, and with ores of moderate and high-grade appears to work to better advantage.

It would appear, in fact, that the limitations of the cyanide process might be summed up as follows:—

1. That it is only entirely successful with free-milling ores, as although it will deal with pyritic ore, it does so at a greatly enhanced cost.
2. That it is inapplicable to ores containing a considerable percentage of coarse gold.
3. That it cannot be economically applied to rich material, as a loss of 2 dwts. in 8 dwts. ore is a very different matter to a loss of say 2 ozs. in 8 ozs. ore.
4. That it is not applicable to ores containing certain metal and mineral combinations.

On the other hand, it appears to be a process admirably adapted for saving gold in the condition of float, *i.e.* in an extremely minute state of subdivision, which cannot be caught by any process of concentration or amalgamation.

* Phillips, *Trans. Am. Inst. Min. E.*, vol. xvii., page 313.

† *Memoranda on the Treatment of Refractory Gold Ores by the MacArthur-Forrest Process.* Wm. Hodge & Co., Glasgow.

Briefly outlined, the MacArthur-Forrest process originally consisted in pulverizing the ore to 40 or 60 mesh size, and then mixing, and agitating it, with a solution of cyanide of potassium (the ordinary standard strength of which was intended to be $1\frac{1}{5}$ parts of cyanide to 100 parts of water by weight).*

After a sufficient time has elapsed for the solution of the precious metals, the leach is transferred to large wooden filter-tanks, the solutions are allowed to settle, and are drawn off (sometimes assisted by pressure or suction), and the gold and silver contained in these solutions as cyanides, are decomposed and precipitated in a metallic condition, by passing the filtrate through metallic zinc turnings. In the course of this treatment the zinc replaces the gold in solution as cyanide of zinc, which dissolves in the water, while the gold is deposited as a dark powder. This is separated by sieving from the undecomposed zinc, when it is found, by testing with chloride of tin, that the whole of the gold has been precipitated from the leach liquor.

The zinc residues are then removed and dissolved in nitric acid, the gold remaining undissolved as a dark brown powder, which is washed, dried, and melted, yielding almost pure gold.

If there is any silver present it is dissolved with the zinc, and can be recovered by adding a solution of common salt, which throws it down as chloride. It has then to be reduced to the metallic state by contact with sheet-iron or zinc, after which it is washed, dried, and melted.

The chemistry of the process† is simple, depending on the affinity of cyanogen for gold and silver, and the ease with which these metals form soluble double cyanides with the alkali metals. The relative affinities of the different metals, according to Mr. Wm. Jones, stand as follows :— 1st gold, 2nd silver, 3rd copper, 4th zinc, lead, arsenic, antimony, etc.

The solvent action on the base-metals can be reduced to a minimum by reducing the strength of the solutions, the readily soluble gold and silver being dissolved out with only traces of copper, zinc, etc.

The best strengths of solutions to use in leaching out the gold from refractory stone depends entirely on the nature of each ore, and it is

* In actual practice, however, these ideas have been modified, and what is known as the percolation system has come into almost universal use, employing standard solutions of a certain strength, first leaching the ore for 6 to 12 hours with 6 to 8 per cent solution, and for 8 to 10 hours more with a weaker liquor, containing 2 to 4 per cent. of cyanide.

† "The MacArthur-Forrest Process for the Treatment of Refractory Gold Ores." *Eng. and Min. Jour.*, New York, 1889, vol. xlviii., page 544.

impossible to lay down any hard-and-fast line. The point must be determined by practical tests.

Filtration of the liquor is accelerated by using a vacuum, and there is no practical difficulty about this, unless there is a large percentage of clayey matter present. The amount of free cyanide in the liquors, after passing through the zinc, is then determined by means of a standard solution of nitrate of silver, and the liquor is thereafter made up to its original strength and used over again.

To extract the gold from refractory ores a number of points must however be observed. If the ore betrays a noted acidity, due to the presence of basic sulphates of iron, etc. (especially marked in the case of disintegrated and weathered sulphides), it should be neutralized with an equivalent quantity of caustic lime in the form of milk of lime. The exact amount of acidity can be determined by shaking up a weighed sample of the ore with water, and adding standard-normal or tenth normal caustic-soda solution till the point of alkalinity is attained, as indicated by litmus or any other suitable tests. The amount of lime is then easily calculated. Some ores show as much as 4 per cent. of acidity in terms of soda, and such ores, on treatment with cyanide solutions without previous treatment with lime, fail to yield their gold contents, whereas when previously treated with lime, the greater part of the gold is easily extracted.

Nearly all sulphides show more or less acidity, but when it is under 0·10 per cent. it may for practical purposes be neglected.

The cyanide solution used should be as free from caustic alkali (NaHO or KHO) as possible, as it is apt to form a sulphide of sodium or potassium with the sulphur of the ores, and thus prevent the gold and silver going into solution. This difficulty, when it does occur, is got over by adding calcium chloride.

The cyanide solutions are best preserved from too great exposure to the air, as a part of the cyanide is apt to be converted by oxidation into the cyanate.

From a chemical point of view it appears, that the economic success of the process, will mainly turn on the price, and consumption of chemicals, and on the time taken in treating large quantities, to extract a given percentage of gold from various combinations of ores in different localities in which the gold exists in a coarse or fine state of division, and in different alloyage, and in which the pyrites is more or less highly concentrated. The ores of the Witwatersrandt seem specially adapted for treatment by this process.

The Russel Process for Silver Ores.

Quoting Mr. C. A. Stetefeldt, on the lixiviation of silver ores, Mr. E. H. Russel, it is said, discovered that a solution of a double salt of cuprous hyposulphite and sodium hyposulphite (formed by mixing sodium hyposulphite with copper sulphate) exert a most energetic dissolving and decomposing action upon metallic silver, silver sulphide, silver minerals belonging to the group of antimonial and arsenical sulphides, and other silver combinations. Hence, if a charge of roasted ore is first lixiviated with ordinary sodium hyposulphite solution to dissolve the silver chloride, and subsequently with cuprous hyposulphite (this solvent is called the extra solution) an additional amount of silver is extracted, which would have been lost in the tailings, by working according to the old method alone. Or, if the roasted ore contain caustic lime and be treated with the extra solution, the deleterious influence of the caustic lime is thereby neutralized. In the same way the extra solution may be applied to extract silver from raw ores without previous chloridizing-roasting, or to lixiviate ores after they have been subjected to an oxidizing-roasting. Mr. Russel also discovered that lead can be completely separated from a sodium hyposulphite solution, as lead carbonate, by sodium carbonate, without precipitating copper or silver.

After decanting the solution from the lead carbonate, silver and copper are obtained from it in the usual way. This method of separating lead prohibits the use of calcium polysulphide as a precipitant for the sulphides, because calcium, entering the regenerated lixiviation-solutions, would also be precipitated as a carbonate with the lead, by sodium carbonate; hence sodium sulphide must be employed. A full investigation has demonstrated that this is by no means detrimental, as sodium sulphide and sodium hyposulphite are more advantageously used in the lixiviation process than the corresponding calcium salts.

Finally, Mr. Russel found that if a hyposulphite solution has a caustic reaction, produced by caustic soda or lime, its solvent power for silver is materially deteriorated. This defect he corrects by neutralizing such a solution with sulphuric acid.

The Van Patera Process (Ordinary Lixiviation).

This process consists in:—(1) Crushing the ore (generally with rolls). (2) Drying the ore. (3) Roasting with salt. (4) Leaching out the base-metals with hot or cold water.* (5) Leaching out the silver with

* The base-metal chlorides in the water may carry off 0·5 to 3 per cent. of silver, but this can be recovered by simple dilution, unless chlorides of lead and antimony be present (which are likewise precipitated), in which case either the ore must be leached from below, under slight pressure, or the silver must be precipitated in troughs outside the tubs.

hyposulphite of soda. (6) Precipitating the silver. (7) Roasting the sulphide of silver and melting to bullion.

It is capable of saving about 85 per cent. of the silver in the chloridized ore, assuming it to be suited to this class of treatment.

The writer will not stop to refer to other wet methods, such as the Claudet and the Kiss, for gold and silver ores, and the Augustine and Ziervogel, used for the treatment of argentiferous copper matte, though they all undoubtedly possess special features of interest.

The difference between the Russel process and ordinary lixiviation, must however be alluded to.

1. It requires a less careful chloridizing-roasting, and on that account a lower percentage of common salt may be used in roasting.
2. It extracts a higher percentage of silver by means of the extra solution. This is specially of importance in treating raw ores, and in lixiviating roasted ores containing caustic lime.
3. It produces sulphides free from lead.
4. It yields lead in the form of lead carbonate as a valuable bye-product.
5. It overcomes the deleterious effect of a caustic lixiviation-solution, by neutralizing it with sulphuric acid.
6. It uses sodium hyposulphite and sulphide exclusively, and not the corresponding calcium salts.

Ores suitable for Treatment by the Russel Process.—It is claimed by Mr. Stetefeldt that almost all silver ores that do not carry a large percentage of lead or copper, can be treated by lixiviation with success and economy. He says :—"I do not mean to create the impression that, from ores containing more or less lead and copper, a high percentage of the silver cannot be extracted by this process. Such ores, however, will in most localities be reduced to better advantage by smelting."

It is safe to state that all ores fit for amalgamation can also be treated by lixiviation, and that the Russel process may succeed where amalgamation is a failure. In cases where lead-bearing silver ores are suitable for concentration, it may be profitable to concentrate the ore by the Krom dry system, obtaining and smelting product, high in lead, and to lixiviate after roasting the tailings and the dust.

The dry system of concentration deserves the preference, because it delivers the tailings and the dust in a condition ready for chloridizing-roasting. In wet concentration, the drying of the tailings would be expensive, and there would be a considerable loss of silver in slimes.

Oxidized ores, containing silver chloride and lead minerals, may be lixiviated after crushing, and the tailings may be concentrated for lead. In this case wet concentration would be most suitable. This has been done at the Old Telegraph mine, Utah. The Russel process is also adapted to the treatment of tailings, resulting from ores which have been worked either by the old lixiviation process, or by amalgamation. Whether it is more profitable to lixiviate an ore raw, or after chloridizing-roasting, or after oxidizing-roasting, must be determined in each case by actual experiment.

Personally, on commercial and other grounds the writer cannot take Mr. Stetefeldt's view with regard to using dry concentration as an adjunct to lixiviation, unless recent improvements have been made in the process, which he is unacquainted with; though, of course, this is merely a side question, and on the main issue Mr. Stetefeldt is no doubt one of our greatest living authorities.

Smelting.

Although this is one of the most important methods of beneficiating ores of gold and silver, such as contain large amounts of lead and copper (a class by themselves), which can rarely be treated in any other way, the writer does not propose going into this branch of the subject, beyond stating that favourable local conditions with regard to nature and grade of the ore, and the possession of cheap and suitable fuel and transport, skilled labour, fluxes, dump-accommodation, water, and other things are necessary to render works of the kind a success. A convenient central situation, and regular supply of ore in sufficient quantity, to keep the furnaces running steadily, being two most important points.

Notwithstanding, however, that the gross cost of a smelting-process is large, the high value of the products it yields frequently justifies its employment, even in comparatively out-of-the-way parts of the world.

Under circumstances of this kind water-jacketed furnaces of the Piltz type, seem to be gaining more and more ground in argentiferous-lead smelting in the Western States of America, and are most highly successful, more particularly as regards cost, which precludes the use of high-class metallurgical fuel (in many places practically unprocurable).

Every metallurgical process gives rise to a certain amount of loss.

The losses in roasting have been already referred to, and some of their causes explained, and as roasting is necessary to prepare most classes of ore* for smelting, it is well to bear the facts in mind.

* If they contain an excess of 5 per cent. of sulphur.

A further loss ensues in the actual smelting, part of the precious and base metals being volatilized or mechanically carried out of the furnace, and parts entangled in the slag, while a still further loss takes place in refining the lead bullion.

As a rule, the loss in smelting proper in the Western States of America, may be put down, the writer believes, at 3 to 4 per cent. of the silver, and a variable amount of lead, depending chiefly on the percentage of the latter metal in the ore.* This loss of lead varies under these and other circumstances from 5 to as much as 15 per cent.

The saving of gold and silver can be put down at 90 to 98 per cent.

The same remarks that apply to concentration, in comparing European and American practice, apply equally to smelting, and the point to determine is how far close work can be carried, to yield most profit on the money invested, in view of the cost of treatment.

The cost of smelting is extremely variable, and depends on the price of labour, fluxes, fuel, composition of the charge, and its behaviour, size of furnace, arrangement of plant, freight and refinery-charges on bullion, and other circumstances.

Under the most favourable conditions as regards ore, fluxes, fuel, etc., the most docile lead ores may be smelted as low as 12s. 6d. to 16s. 8d. per dry ton. Under unfavourable conditions the cost of smelting may run from £1 0s. 10d. to £6 5s., but from £3 2s. 6d. to £5 4s. 2d. is a closer average. In some cases, in dealing with smelting ores it will pay better to concentrate close, and ship and sell the concentrated ore, than to smelt on the spot. In this connexion, it is to be noticed that the degree to which concentration ought to be carried, in preparing an ore for smelting, depends on the value or otherwise of the concentrates and the gangue as a flux. The relative cost of shipping the work-lead to refining works, or of refining locally, has also to be considered.

By far the greater proportion of the ores produced in the silver-lead mining camps of the States are sold under contract or at public auction, on account of the mine owners, to custom smelting establishments, to which sampling works for mechanically sampling the ore are generally attached.

The dry weight, *i.e.*, the percentage of moisture, is usually left to the smelter to determine.

A higher charge is necessarily exacted when the ores are sulphides or highly siliceous. Zinc, sulphate of baryta, and occasionally silica, are

* "Notes on Western Lead Smelting," W. S. Keyes. *Eighth Report of the California State Mining Bureau*, page 804.

charged for at so much per unit, in excess of a certain percentage fixed by the smelter; 5 per cent. more zinc being commonly allowed in oxidized than in sulphuretted ores.

When the ore is calcareous, or highly charged with oxide of iron, a rebate on the ordinary public tariff-price, which represents their maximum charges, is often allowed by smelters. As regards the lead, some smelters require a certain minimum, and an extra charge is made for each per cent. of lead below it; as a rule, no payment is made when the assay shows less than 10 per cent.

It is usual to deduct 5 per cent. for loss of silver in the ore, and calculate the value on New York quotations for the balance; when, however, the ores are dry or siliceous, a still further deduction is made, often as high as 10 per cent. for 100 ounces ore, but less for higher grades.

The gold is paid for from $\frac{1}{10}$th of an ounce per ton upwards, at the rate of £3 15s. to £3 19s. 2d. per ounce. New York quotations for silver and lead on the day of settlement, are assumed as the basis of price.

(To be continued.)

NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,
HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE,
FEBRUARY 11TH, 1893.

MR. J. B. SIMPSON, PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the last General Meeting, and reported the proceedings of the Council at their meetings on January 28th and February 11th.

The following gentlemen were elected, having been previously nominated :—

MEMBERS—

Mr. THOMAS ALEXANDER ALLAN, Consulting Mining Engineer, Brightlands, Richmond Hill, Surrey.

Mr. THOMAS BREAKELL, Mining Engineer, 92, Denmark Road, Greenheys, Manchester.

Mr. WALLACE THORNEYCROFT, Mining Engineer, 6, Dixon Street, Glasgow.

Mr. HENRY THOMAS WALES, Mining Engineer, 104, St. Mary Street, Cardiff.

ASSOCIATE MEMBERS—

Mr. JAMES ERNEST BLENKIRON, Mining Engineer, West House, Arkengarthdale, Richmond, Yorkshire.

Mr. J. CARTMEL RIDLEY, Steel Manufacturer, 3, Summerhill Grove, Newcastle-upon-Tyne.

ASSOCIATE—

Mr. JOHN SPRY, Surveyor, Murton Colliery, *via* Sunderland.

The following gentlemen were nominated for election :—

MEMBERS—

Mr. M. W. BARKER, P.O. Box 1,463, Johannesburg, Transvaal.

Mr. HENRY B. BUNKELL, P. O. Box 962, Johannesburg, Transvaal.

Mr. WILLIAM YOUNG CAMPBELL, Mining Engineer, etc., P.O. Box 357, Johannesburg, Transvaal.

Mr. William Charlton, Mining Engineer, Linares, Provincia de Jaen, Spain.

Mr. Sherard Osborn Cowper-Coles, Manager, London Metallurgical Company, Limited, 80, Turnmill Street, London, E.C.

Mr. James Hubert Dangar, Camborne, Cornwall.

Mr. J. Hubert Davies, Electrical Engineer, P.O. Box 455, Johannesburg, Transvaal.

Mr. Alfred Dumat, Mechanical and Railway Engineer, 1, Loop Street, Pietermaritzburg, Natal.

Mr. E. W. Evans, Mining Engineer, P.O. Box 357, Johannesburg, Transvaal.

Mr. Louis Faragher, Mine Agent, Cape Copper Co., Ld., Ookiep, Namaqualand, South Africa.

Mr. Albert H. Halder, Consulting Engineer, Pietersburg, Transvaal.

Mr. J. Kerr Houlgate, Mining Engineer, 69, Lowther Street, Whitehaven.

Mr. John Arthur Jones, Mining Engineer and Analytical Chemist, Gijon, Asturias, Spain.

Mr. John O'Gorman Lalor, Mining Engineer and Metallurgist, P.O. Box 357, Johannesburg, Transvaal.

Mr. Henry Lakin Lawrence, Mining Engineer, Mines de Pierrefitte, Pierrefitte-Nestalas, Hautes Pyrénées, France.

Mr. Richard A. D. Liebert, Mechanical Engineer, P.O. Box 357, Johannesburg, Transvaal.

Mr. R. H. Longbotham, Mining Engineer, Westgate, Wakefield.

Mr. David Alexander Louis, Metallurgical and Mining Engineer, 77, Shirland Gardens, London, W.

Mr. John S. MacArthur, Metallurgist, 12, Knowe Terrace, Pollokshields, Glasgow.

Mr. John Hodgson Nicholson, Mining Engineer, Seaton Delaval Colliery, Northumberland.

Mr. Francis Daniel Peter, Mining Engineer, Greymouth, New Zealand.

Mr. John Gunsen Selkirk, Mining Engineer, Dalton-in-Furness.

Mr. A. Humboldt Sexton, Professor of Metallurgy, Metallurgical Department, Andersonian Buildings, 204, George Street, Glasgow.

Mr. Charles Liddell Simpson, Mechanical Engineer, Engine Works, Grosvenor Road, Pimlico, London.

Mr. Arthur James Stevens, Mechanical Engineer, Uskside Ironworks, Newport, Monmouthshire.

Mr. W. Henwood Trelease, Chief Agent of the Pestarena United G. M. Co., Ld., Pestarena, Vall'Anzasca, Novara, Italy.

Associate—

Mr. Joseph Thompson, Overman, 38, Church Street, Murton Colliery.

Mr. Robert T. Moore read the following paper on "Joseph Moore's Hydraulic Pumping Arrangement":—

JOSEPH MOORE'S HYDRAULIC PUMPING ARRANGEMENT.

By ROBT. THOS. MOORE.

The problem of how best to drain our mines is one of the greatest interest to an institution such as this. It would be easy to dilate upon the inconvenience of pump-rods and upon the expensive nature of their up-keep, and also upon the destructive and expensive nature of break-downs, when the full power of the engine is continued without resistance. This, however, is so well known to the members that it is unnecessary to take up the time by discussing it.

But it may be stated that these difficulties have, in a great measure, set men thinking as to whether they could not dispense with pump-rods altogether ; and the direction that this search for a new method so far has taken has been to place the steam engine at the bottom of the pit or in the mine, carrying the steam from the surface or erecting boilers underground. Within the last few years electricity, hydraulic power, and other expedients have been tried with varying degrees of success.

The arrangement now to be described is a form of hydraulic pumping. It is at present extensively used in Scottish mines, and it is thought that a description of it may be of interest to members of the Institute.

The arrangement enables two columns of water to be substituted for the ordinary pump-rods connecting the steam engine with the pump, which may be placed any distance from each other. Advantage is taken of the fact that water is practically incompressible, and that if a pipe of any length is filled with water a plunger at one end will, if pushed forward, propel a plunger placed at the other end in the same direction. If there were two pipes and two plungers there would be a reciprocating motion.

The action will readily be seen from Fig. 1, Plate XVII., which is not drawn to scale, but is merely for the purpose of illustrating the arrangement.

At the steam engine is a hydraulic cylinder *a b*, in which works a piston *p* called the power ram, and this is moved by a crank driven by the steam engine.

At the pump is a cylinder *c d*, exactly the same as that at the steam engine and in this is a piston *q* also called a power-ram. This power-ram is attached to the plunger *r* of a double-acting pump.

A pipe *e* connects the end *a* of the power-cylinder at the engine with one end *d* of the power-cylinder at the pump. A similar pipe *f* connects the other end *b* of the power-cylinder at the engine with the end *c* of the cylinder at the pump.

When both cylinders and pipes are filled with water, and the piston *p* is moved by the engine from *a* towards *b*, the water in the cylinder is forced through the pipe *f* into the *c* end of the cylinder *c d* at the pump, and the power-piston *q* is moved from *c* towards *d*.

In like manner when the piston at the engine moves from *b* towards *a*, that at the pump moves from *d* towards *c*, carrying the pump with it. In this way the pump is moved backwards and forwards in the same way as if there had been a direct connexion with rods between the steam engine and the pump.

In order to provide for leakages a tank is placed above the level of the highest point of the power-pipes with pipes from it to the power-pipes *e f*. On the end of each pipe is a valve opening inwards, so that when the engine moves the power-ram *p* from *b* to *a*, water may run from the tank into the pipe *f*. When it moves from *a* to *b*, water may run into *e*.

An arrangement of this kind will not work satisfactorily unless there are means for regulating the stroke of the pump. For example, if by any cause the piston when being moved from *d* to *c* were to stop before completing its stroke the succeeding stroke from *c* to *d* would be too long, and the ram would be driven against the end of the pump and do damage. To obviate this, a case with two small valves *j k* is placed at the power-rams with pipes *g h* connecting the power-pipes *e* and *f*. Attached to these valves are levers *l* and *m* which work thus:—

When the piston passes the ordinary stroke, the tappet *s* strikes the lever *m* which opens the valve *j*, and the compressed water passes from the column *f* through the opened valve *j* lifting and passing through the valve *k* and running into the column *e*, thus taking the pressure off the power-piston and stopping it. In the next stroke from *d* to *c* the water is stopped by the valve *k*, and the piston makes the full stroke, stopping before the tappet reaches the lever.

It will thus be seen that the adjustments are automatic, and in practice they are found to be very complete.

Having described the principle of the arrangement, the following

description of a plant at the Linlithgow Oil Company's Champfleurie mines, will show an application of it.

The steam engine (Figs. 2 and 3, Plate XVII.), is placed on the surface, 50 feet from the mouth of the mine which dips from the surface at an inclination of 1 in 3: the pump is placed 500 yards from the surface and throws the water 83 fathoms vertically.

The engine is high-pressure, with a cylinder 20 inches in diameter and 4 feet stroke, and is geared to the pumping crank-shaft at the rate of 6 to 1, the small wheel being 20 inches in diameter and the large wheel 10 feet in diameter. The pump is double-acting, 12 inches in diameter and 4 feet stroke. It is driven 8, 4 feet strokes, and discharges 300 gallons per minute. It has been in operation for two years.

The surface power-rams are two solid steel plungers *b* and *c*, 6 inches in diameter and 4 feet stroke, and are worked by a connecting-rod *d* from the crank *e*. Malleable iron power-pipes *f* and *g*, $2\frac{3}{4}$ inches bore and $\frac{3}{8}$ inch thick, pass from the ends of the power-ram casings to the power-rams at the pump. A pipe *j*, $\frac{3}{4}$ inch in diameter reaches from the supply tank to the valve-box *k*, with two valves 1 inch in diameter opening inwards to the power-pipes. A pipe *l*, $\frac{3}{4}$ inch in diameter connects the top of one valve with the power-pipe *f*, and another *w* connects the other with the power-pipe *g* at the pump.

The mine pump (Figs. 4 and 5, Plate XVII.) is horizontal and consists of two rams *m* and *n*, 12 inches in diameter and 4 feet stroke, face to face, and connected to a cross-head *s*. The power-rams *o o* and *p p*, each 4 inches diameter, are placed alongside of each pump-plunger, and are connected to the cross-head.

The power-rams *o o* are connected with each other by the pipe *q*, and are connected with the surface power-ram by the power-pipe.

In like manner the power-rams *p p* are connected together and connected with the surface power-ram by the power-pipe.

The tappet *s* on the cross-head is so placed that when the pump has come to near one end it comes into contact with the levers *t* or *u*, and one of the equalizing valves in the casting *v* is opened, and the water passes from one power-pipe to the other, and the plunger stops.

The maximum pressure of the water in the power-pipes is about 2,000 lbs. per square inch.

At Balquhatstone colliery, Slamannan, there are two sets of double-acting 12 inches pumps worked by one geared engine, raising 600 gallons a minute from the bottom of a vertical shaft 65 fathoms deep. The

driving cranks for the surface power-rams are fixed at right angles, so that the engine is balanced. Either pump can be worked independently of the other by disconnecting the power-rams at the engine, so that it forms a duplicate plant. This is found to be a very good arrangement and several installations have been fitted up on this plan.

This system has been applied for sinking. Figs. 6, 7, 8, and 9, Plate XVIII., show a sinking pump which was used for raising 300 to 500 gallons per minute during the sinking of a shaft 100 fathoms deep.

It will be seen from the drawing that the pumps slide in wooden guides like a cage, and the whole arrangement of pumps, power-pipes, and rising column is carried by iron rods from a hydraulic ram fixed over the pit. The plungers, which are hollow, are placed vertically, and the one plunger is half the area of the other, so that the pump is double-acting. The lower pump plunger is 18 inches in diameter and has a clack inside of it. At the bottom of the plunger-casing is a clack-piece, and below that is the suction.

The upper plunger is $12\frac{3}{4}$ inches in diameter.

The upper casing is connected to the rising main, and the lower one is connected with the upper by four bolts.

The power-rams work in the same way as in the horizontal pump already described.

The following is the action of the pump:—When the plungers ascend, the lower plunger lifts a quantity of water represented by a plunger 18 inches in diameter and it passes up through the lower valve, the valve inside the plunger remaining shut. When the plungers descend, the valve at the bottom shuts, while the valve in the plunger opens, allowing the whole water from the 18 inches plunger to pass through it, but as the 12 inches plunger is descending out of the casing at the same moment, only half of the water ascends to the surface and the other half fills the space occupied by the 12 inches plunger. When the plunger ascends again, the 12 inches plunger discharges the water into the rising main.

Immediately above the upper barrel is an air-vessel, 14 feet long, made of malleable iron. The water passes up a central pipe which dips down from the top of the air-vessel, and it is the annular space round this central pipe which forms the air-space, the central pipe communicating at the top with the rising main.

A cup and small discharge-pipe are provided for receiving and at convenient intervals discharging any sediment deposited from the ascension pipe.

The rising main is made of malleable iron, in 24 feet lengths, and having cast-iron flanges riveted to them. The joints are spigot and faucet, and are made with a ring of gutta-percha, $\frac{3}{8}$ inch in diameter.

The hydraulic ram is worked with a small steam engine, and raises and lowers the whole pumps and columns. The lever which works this engine stops the main engine on the surface, and opens a valve communicating with the power-pipes, so as to let off the pressure due to their sliding into the sleeve-pipes when the pump is raised.

The upper two lengths of the rods are provided with holes into which a key is put for the plunger of the ram to press against.

Attached to the girder which supports the ram are two rods *a a* (Figs. 10 and 11, Plate XVIII.) of the same strength as the main rods, carrying a cross-head *b*, with recesses like a boring tool for catching the heads of the rods, which have a shoulder made to fit it. When the ram has run its full length this is put under the rods, and the whole weight of pipes, etc., rested upon it. A new length of rods is put in, and the ram is run out until it comes in contact with the key. The temporary rest is then removed and the ram is ready for further lowering.

On the top of the rising column is placed a "hogger" of sufficient length to allow the water to fall into the discharge drain, and the power-pipes work in stuffing boxes *l m*.

The arrangement for changing the upper clack is rather peculiar. The smaller plunger is carried down into the middle of the larger one as shown in the drawing, and at Fig. 14, Plate XVIII., and terminates in a recess in which works the T-shaped head of the clack. When it is in operation the lid has free room to move up and down without check. When the clack is to be shifted the column of water is run off, the pump raised to the top of its stroke, and the upper plunger disconnected from the lower one and suspended by two links. The lower plunger is then lowered down. The clack-lid remains hung by the T-head on to the inner plunger; and as the lower plunger descends is left outside, when it is drawn out of the slot and renewed. The lower plunger is then raised and the two connected, the clack having then room to work as before.

This sinking-pump was erected in a gold-mine in Nevada, but so far all the pumps erected in this country have been horizontal, and fixed either at the bottom of the shaft or in the workings.

The steam engines for driving these pumps may be either direct-acting or geared, as may be found most convenient. In most of those erected, the engines are geared in the proportion of 6 to 1, so as not to drive the power-rams more than 8 strokes per minute.

A pump worked on this system gave a useful effect of 68 per cent. of the indicated diagram of the steam engine with an inclined distance of 800 yards between the engine and pump.

Another pump gave over 66 per cent. of efficiency. These results compare very favourably with electricity and compressed air.

Some of the advantages of the system are :—The large percentage of useful effect got from it. The engine and pumps can be put away from the shaft, leaving the pit-head and pit-bottom unobstructed and the shafts clear of any moving pitwork. The saving of room will be apparent when it is stated that with the Balquhatstone pump there are only four lines of $3\frac{1}{2}$ inches pipes in the shaft to drive pumps throwing 600 gallons of water a minute, 65 fathoms vertical. In inclined and dip workings the power-pipes may be led round any number of bends. The parts underground are of the simplest description and not liable to get out of order, and there is no need for an attendant.

Some of the pumps have worked covered with water. Another advantage, and a very great one, is that the small amount of room taken up by the apparatus facilitates the erection of duplicate arrangements, whereby, should any breakdown take place, the pumping could be continued by one of the engines until the repairs are completed.

As regards cost, perhaps the simplest way will be to quote an offer made to supply a colliery with plant for raising 400 gallons of water per minute from the bottom of a shaft 150 fathoms deep. Boiler pressure 100 lbs. per square inch.

The water to be raised in two 75 fathoms lifts, a double-acting pump with plungers 14 inches in diameter, 5 feet stroke, to be put at the pit bottom to throw half-way up the shaft, at which point another pump of exactly the same size is placed to throw the water to the surface.

Engine—		
Compound high-pressure cylinder, 18 inches in diameter Low-pressure cylinder, 30 inches in diameter ... ... Stroke, 5 feet ... ... Gearing, 6 to 1, large wheel being 12 feet in diameter	...	£900
Two sets of pumps each.		
Surface power-rams, $6\frac{3}{4}$ inches in diameter ... ... Stroke, 5 feet ... ... Pumps, double-acting, 14 inches in diameter ... ... Power-rams (two pairs each), $4\frac{1}{2}$ inches in diameter... Stroke, 5 feet	...	£1,100
Power-pipes, $3\frac{1}{2}$ inches bore, $\frac{7}{16}$ inch thick, say 3,000 feet at 3s. 6d.		£525
Total cost		£2,525

Mr. C. C. Leach asked what advantages the Moore pump had as compared with the ordinary hydraulic pump?

Mr. R. T. Moore said one advantage was that the water in the pipes was never changed, and consequently there was no difficulty as to working with clean water. Again, in the ordinary hydraulic pump some form of valve was required to admit the water alternately into each end of the power-cylinder, while in this pump there were no valves; and the arrangements underground were very much simpler than was possible with the ordinary hydraulic pump, which was practically an engine working underground.

Mr. H. Lawrence said one great advantage accruing from this system of pumping was that it was very suitable for placing at the bottom of a shaft where there was liability of being drowned out, because it would still work unless the valve arrangement went wrong. The system of the ram pump was not in any way different from what was otherwise in use at the present time. The novelty consisted, to a large extent, in having two pipes filled with water, which enabled them to lead the power whither they liked, but he thought there must be a large loss by friction in the moving of the water in the pipes. They all knew that it took a very much larger power to put a body in motion than to keep it in motion, and, although the writer stated that he got 68 per cent. of useful effect, that was not a very high percentage for an economical pump. Then, again, in such instances as the new pumping arrangements now being erected at the Wallsend collieries, he was afraid that as an engineer he could not advise its use for a large pumping-plant, but for small depths, and for the uses referred to in the paper he dared say it would work very well. As to the saving of space in the shaft, he did not know where the special advantage arose in this system, inasmuch as two pipes were put down the pit for the motor, but there must be another, which was not shown, for the delivery of water to bank. So that they must have three, nay, he thought he had even seen four pipes; and if they placed an ordinary direct-acting pump down the pit they would only have one pipe, which, he thought, was an advantage, as it would have very much less friction to contend with. The system described was deemed to be very applicable to sinking shafts, because it could be moved up and down in stages as the water was lowered, and the applications mentioned in the paper verified that assumption. There was another point, but he supposed they were all subject to that, viz., the question of very deep pits, where 300 or 400 lbs. per square inch would represent the pressure on these pipes, whether in such cases a very much larger amount of repair would not be required;

by having so many pipes there would be much greater risk of breakages, and he did not grasp exactly how Mr. Moore proposed to work with one pump in case anything went wrong with one of the pipes.

Mr. W. C. Blackett said there were several points which appeared to require a little elucidation. The writer opened the paper by saying it would be easy to dilate upon the inconvenience of pump-rods, and upon the great cost of their maintenance, and also upon the destructive and expensive nature of any accident; he then proceeded to describe the Moore hydraulic pump, thus suggesting that it was intended to do away with pump-rods, and at the same time to have equally efficient apparatus, but later in the paper he mentioned that a pump working on this system gave a useful effect of 68 per cent.; was it meant to infer that this system, with 68 per cent., was superior to the ordinary pump-rods in useful effect?

Mr. A. L. Steavenson said he had received from Mr. Joseph Moore copies of papers which he had read descriptive of his invention. It appeared to be a very useful pump, and of very good design; in fact, he had referred to it in the paper he recently read before this Institute on the subject of the different methods of power transmission. For pumping out of a shaft, however, he should still adhere to the old fashion; but there were many collieries where they could not use the ordinary pumping-plant with spears in the shaft, and in such cases nothing could possibly be better than Mr. Moore's arrangement, especially if they knew the volume of water to be pumped, and that the feeder did not increase or alter from time to time. In these cases it would be difficult to increase or duplicate the pumping-plant. The appliance seemed most ingenious, and under certain circumstances would do work which no other appliance could perform. He suggested that the naming of one or two parts of the pump might be altered. Reference was made to the piston or ram—but a piston is not a ram. He would have preferred to call it ram or plunger, for if they used the words indiscriminately they must add double-acting, whereas, if they kept them distinct one meant single and the other double-acting. Had the writer at any time tested the useful effect? Had he measured the water pumped as compared with the work done by the steam in the engine?

Mr. R. T. Moore said the work done in raising water (as measured by the height the water was lifted and the water pumped) was 68 per cent. of the indicated horse-power of the steam engine. With regard to the alternate wording ram or piston, this was only meant to convey that either might be used.

The PRESIDENT said he would draw attention to one point, with respect to placing the engine on the surface for pumping instead of adopting the ordinary method. The useful effect got out of the Moore engine appeared to be only 68 per cent., but with ordinary pumping-engines the efficiency was at least 80 per cent. If they were to substitute a pumping-engine with 68 for one yielding 80 per cent., although probably the first cost (according to the figures given in the paper) might be less, yet the annual cost of fuel would be larger—probably nearly 20 per cent. more in the case of the hydraulic engine than with the ordinary pumping-engine. There was no doubt, in certain cases underground, the use of the Moore pumping-plant would prove very advantageous, and it seemed to be extremely ingenious; but he would very much hesitate to adopt it on the surface in preference to ordinary methods, on account of its low useful effect.

Mr. H. LAWRENCE referred to some experiments he made many years ago with a beam pumping-engine, which yielded 83 per cent. of useful effect.

Mr. R. T. MOORE agreed with the President that the direct-acting engine placed over the pit would give an efficiency of 80 per cent.; he had himself indicated as high as 85 per cent.; but it must be remembered that the direct-acting engine, pulling directly on the rods and lifting the plunger, was very different from the case in which the Moore hydraulic pump obtained 68 per cent., the pump being placed 800 yards down a mine with varying inclinations. Had the engine been fixed on the surface with the pump at the bottom of the shaft, the useful effect would no doubt have been much higher. He was prepared to agree with the President that a higher useful effect would be got by placing the steam cylinder directly over the pit and pulling directly on the plunger than with any arrangement such as that described in the paper or with electricity, but arrangements such as he had described had many advantages to commend them. There was the advantage of being able to place the steam engine at a distance from the pit and leaving the ground clear, and there were no moving parts in the shaft. Mr. Blackett had mentioned the questions of breakdowns with pump-rods, but he thought that with this system there would be no breakdowns in the shaft. The only moving parts were at the engine on the surface and at the pump underground; if the power-pipes were put in of sufficient strength, as there was no difficulty in doing, the repairs cost practically nothing. The loss by the friction of the water in the pipes was a very small matter. It was represented by a column of water $2\frac{1}{2}$ per cent. of

the length of the pipes. With 4,000 feet of pipes, this was 100 feet of water or 43 lbs. per square inch, and this with an effective pressure of 1,000 lbs. per square inch was under $4\frac{1}{2}$ per cent. As a matter of fact, in experiments which he had made there was not a great loss by friction in the pipes. It certainly represented much less fuel than the loss by condensation in a long range of steam pipes. One gentleman said that it was necessary to take only one pipe down the shaft in the case of steam pumps, but it must be borne in mind that in such cases they were using a fluid (steam) at 80 lbs. per square inch, whereas in the hydraulic pump they were working with 1,000 lbs., consequently a smaller pipe was needed at the higher pressure, and although two pipes were required the room taken by the two was not so great as by the one for steam. He thought that a steam engine underground was never so well looked after as one on the surface, and the tendency nowadays was to keep steam out of the pits as much as possible. With regard to duplicating pumps, those he referred to in the paper were simply two separate sets of pumps with two separate sets of power-pipes. Either set could be worked by disconnecting the pipes at the surface. They were really two entirely separate and distinct pumps.

Mr. H. Lawrence asked whether, supposing they were working at 40 or 50 fathoms and one of the pipes burst, they could work one-half of the pump while repairing the pipe?

Mr. R. T. Moore—With this plant they could. When a valve was being changed in one of the pumps, one set of power rams on the surface was disconnected and the other kept going. As a matter of fact they had very little trouble with burst pipes; they were tested to two tons per square inch before being used.

Mr. C. C. Leach asked if there was any trouble with the water in the case of frost?

Mr. R. T. Moore said the joints were the ordinary hydraulic pipe coupling, tapering to the centre. They had had no trouble with the joints, and frost did not affect the water in the power-pipes while the pump was going. Pipes in exposed situations were covered with non-conducting material to prevent freezing when the pump was standing.

Mr. F. Berkley asked if either the sinking or the horizontal pump required much fixing?

Mr. R. T. Moore said they did not.

Mr. Mulholland asked if the length of stroke was the same in the pit as at the engine? It seemed to him that there would be some efficiency lost in the adjusting arrangements below.

Mr. R. T. Moore replied that, so far as the adjustment by the valves was concerned, this only took place occasionally and only when there was some leakage. This might be not more than one stroke out of 100. There no doubt was loss on the particular stroke when it did touch the valves, but it was inconsiderable. With regard to the difference between the stroke on the surface and underground, although water was practically incompressible, it was not altogether so, and with this system, if there was a long length of pipes, the compression was such as to make the stroke underground rather shorter than on the surface; but this difficulty was overcome by making the plunger on the surface a little larger than the other.

Mr. W. C. Blackett said that this pump appeared for many in-bye purposes to be an extremely good one and probably one of the best, but he did not see that its use offered any advantage when placed at the pit bottom or in sinking pits. The writer claimed certain advantages by not having any moving machinery in the shaft, but there was the moving columns of water, and he did not see that this system possessed any advantages over steam. It appeared to him, too, that by putting steam down the shaft they would get a higher useful effect than either 68 or 70 per cent. The arrangement was fairly on all fours with compressed air, except that it gave a much higher efficiency. The writer mentioned that steam engines were not so well looked after underground as on the surface, but he (Mr. Blackett) took exception to this statement. He believed that in many cases they were quite as well looked after, but supposing they were not, that fact hardly constituted an advantage in favour of another system, inasmuch as the hydraulic motor which Mr. Moore would put down the pit was also an engine. However, the arrangement was an excellent one for certain positions in-bye, and if he was now in order he would move a vote of thanks to the author for his excellent paper.

Mr. R. T. Moore said this was not an underground engine. It was no more complicated than an ordinary ram-pump, and did not require any more attention. He would like to draw attention to the fact that rods, even in a vertical shaft, did not give much better results than those he had got with a long length of pipes and varying inclinations; had rods been used in that case they would not have given nearly such a good effect. One advantage—he did not know whether he had made it quite clear—was that they could put the most economical form of rotary engine to drive the pumps, and he was inclined to think that, with high-speed triple-expansion engines, as good economical results could be obtained

with this system as by any direct-acting system. Steam pumps with the engine at one side and the ram at the other were wasteful of steam, and with a carefully constructed rotary engine on the surface one would get better results with this system than with a steam pump down the pit. There was always loss by condensation in taking the steam down the pit, and he was satisfied that the coal burned for raising a given quantity of water was much greater than with the hydraulic system at the Shotts Iron Co.'s Mauricewood mines, when hydraulic pumps were put in to replace steam pumps driven by steam taken down the pit and dip mines. The manager of the colliery stated that the monthly consumption of coal was reduced from 859 to 567 tons.

The PRESIDENT said it would be very interesting to know the useful effect of the engine at Balquhatstone colliery if it could be ascertained.

Mr. MOORE said he would obtain this information.

The PRESIDENT believed it would be the pleasure of the meeting to carry the vote of thanks proposed by Mr. Blackett. He was sure the paper had given them something to consider very seriously, and no doubt there was a great deal more in the invention than they already appreciated.

The motion was cordially adopted, and Mr. R. T. MOORE briefly acknowledged the compliment.

The SECRETARY read the following paper by Mr. Bennett H. Brough on the "Use of Cement in Shaft-sinking":—

USE OF CEMENT IN SHAFT-SINKING.

By BENNETT H. BROUGH.

Of late years there has been noticeable a distinct increase in the boldness of the application of Portland cement to engineering works. The advantages of substituting this material for cast-iron and timber are obvious. It neither corrodes nor decays; in damp situations its strength increases with age; and it is not appreciably subject to contraction and expansion.

In shaft-sinking cement is found a valuable auxiliary, and in the special setting of masonry, known as coffering, it is largely used for laying the rings of brickwork. Cement, too, has recently been employed in an ingenious manner on the Continent for consolidating shifting sand in water-bearing strata. The method employed consists in injecting powdered cement, by means of compressed air, steam, or water under pressure, into the ground to be consolidated. The cement is screened in order to free it from lumps, and the powder is taken by an injector which forces it through a flexible pipe into a perforated tube sunk in the soil to the required depth. In this manner the soil becomes impregnated with the powdered cement, whilst the water is driven away before it.

A more recent Continental development of the application of cement is the use of tubbing of cement-blocks for stopping back water where the strata contain large quantities. This process has been found to offer great advantages in cases where the pressure is excessive. The cement-blocks used are so shaped that they can be made perfectly water-tight. Their form, too, admits of easy setting. The shape of the blocks will be apparent from Figs. 1, 2, and 3, Plate XIX. They are segmental, and are provided on the lower side with a semi-circular groove, which corresponds with a semi-circular ridge on the upper side, the groove being slightly deeper than the corresponding ridge, so that space is available for the reception of cement-mortar. On each side of the block there is a hollowed-out channel for the reception of concrete, the weight of the block being in this way somewhat reduced. In each block two holes are cored out, one of which is central and vertical, whilst the other passes radially from front to back, not in the middle of the block, but at some distance below that point, and intersects the vertical hole. The height of a block is 30 to 40 inches, and its weight 1,543 to 1,763 pounds.

In tubbing a shaft with cement-blocks, the first layer of segments will be laid on the curb, and on this a second layer, and so on, the vertical joints being broken in each course. In order to place a segment in position, in the vertical hole a rod is placed which is provided with a ring or handle at the top and with a slot near the lower end. (Fig. 4, Plate XIX.) This rod is pushed down until the slot is opposite the horizontal hole, and then a bar (Fig. 5, Plate XIX.) is pushed through, until it passes through the slot and holds the rod fast in the block. The winding-rope may then be connected with the ring of the rod, and the block lowered into position. Previously, however, the trough in the block below is filled with cement-mortar. The ridge of the block when lowered coming in contact with the mortar forces it into every crack, and a water-tight connexion is effected between the horizontal layers of blocks. The vertical hole is then filled with a lump of clay which is rammed in, and a bar is pushed through the horizontal hole which has become choked at the point of intersection, and thus the whole is rendered free again. The rod is allowed to stay until the clay has hardened. On withdrawing it a channel is formed, through which the water collecting behind the tubbing can flow. Between every two blocks a tubular joint is formed, which is filled with cement-mortar or concrete rammed in as tightly as possible. The horizontal holes, which had hitherto served as drainage-channels, are closed by cork, wood, or clay. And if at a certain height in the shaft there is a supply of good water it may without difficulty be piped away.

This method has been successfully applied at the Cecilia ventilating shaft of the Serlo colliery, belonging to the Prussian Government, at Saarbrücken. The cement segments in this case measured 15·75 inches in thickness and 23·62 inches in height and in length. The shaft is 9 feet 10 inches in internal diameter. The bed having been formed and the wedging-curb placed in position, the laying of the cement segments was at once proceeded with. When the segments were formed into a complete ring, the perpendicular grooves were filled in with a special mixture of cement, and concrete was rammed in the space between the segments and the sides of the shaft. Above each cement block a thin layer of cement was spread, and on this the segments of the ring above were placed (Figs. 6 and 7, Plate XIX.).

In other cases a double ring of cement segments has been employed (Figs. 8 and 9, Plate XIX.). Here a course of cement blocks for the back wall is first laid, and concrete rammed in behind. This course is about half a block higher than the inner layer, so as to break the horizontal joints. The courses of the inner ring are then laid in such a

way that the vertical joints do not coincide with those of the back ring. The joints are carefully filled with cement, and the intermediate space, of about 4 inches, between the inner and outer rings is then filled with concrete tightly rammed in. In this manner a thoroughly water-tight tubbing is obtained.

The tubbing of a shaft 500 yards in depth and 17 feet 2½ inches in diameter, at the celebrated potash-salt mines of Leopoldshall, near Stassfurt, in Prussian Saxony, was constructed in this manner, in order to prevent the admission of the enormous volumes of water met with in the saliferous strata. Several shafts, too, in the Saar coal-field have been made on this system. The process has been patented in this country. (British patent, 1891, No. 12,145.)

The cost of putting in cement tubbing appears to be considerably less than that of the ordinary brick or iron tubbing. The cost of cement tubbing with a single ring of segments is 25s. 6d. per cubic yard. This estimate includes the cost of material and labour for making the blocks as well as the setting and filling with concrete. Thus, a ring of 13 feet 1½ inches in internal diameter and 13·78 inches in thickness, containing 183·65 cubic feet costs £7 7s. per yard. In the Bildstock shaft at Saarbrücken, which when complete will be 500 yards in depth, 197 feet of cement tubbing of these dimensions has been put in at a contract price of £12 17s. per yard.

In order to compare the cost of cement tubbing with that of brick and iron, the case may be cited of the cost of tubbing a shaft 17 feet 2½ inches in diameter, where large quantities of water had to be dealt with. At the prices current in Germany the cost of this would be as follows:—

	Per Yard.
1. Iron	£78
2. Brick	54
3. Cement	24

In the last case the tubbing consisted of a double ring of cement blocks, 13·78 inches and 9·84 inches thick respectively, with the 7·08 inches of intermediate isolating space, and the space behind the tubbing filled in with concrete. The estimate includes the royalty of 27s. per yard paid to the German company owning the patents. Cement tubbing of these dimensions was put in at the mines of Leopoldshall at a contract price of £21 4s. exclusive of royalty.

The process has not been in use long enough for any data to be available as to the durability of cement tubbing. So far, however, the tubbing put in has stood wonderfully well. The process is one full of

promise, and its development should be watched with interest. A great point in its favour is the simplicity of the design of the moulded cement blocks which should enable them to be made, without special machinery, on the spot where they are required to be used.

Mr. STEAVENSON observed that the block system had been patented, although so long as twenty-five years ago shafts had been lined with them. If there was a fissure in the side of the shaft and plenty of water flowing out of it, he did not think it would be possible to get liquid cement to set in the shaft. The particular process by which this was done might be patented, but the mere fact of using cement in the shaft was not new.

The PRESIDENT said that in 1861 Mr. Watson communicated a paper* on the subject of cement walling, in which a similar block system was explained, and in which an estimate of the cost was given showing the difference as compared with iron, brick, and cement, but he did not think they could go into this matter unless they had more detail as to the respective costs of material and labour.

Mr. W. C. BLACKETT asked for further particulars of the cost of £54 attributed to brick as compared with £24 for cement under similar circumstances?

Mr. M. WALTON BROWN said probably the most interesting of the systems referred to in Mr. Brough's paper was that briefly dealt with in the second paragraph, the use of injections of cement for consolidating quicksand or water-bearing strata. In the case of cement blocks, the writer did not refer to the increased cost of sinking the shaft at Leopoldshall with a rock diameter of at least 23 feet, so as to leave after insertion of the cement blocks, etc., a finished shaft with an internal diameter of 17 feet.

Mr. F. BERKLEY said one advantage of cement tubbing appeared to be overlooked, and that was it would not corrode like iron.

Mr. M. WALTON BROWN said the late Mr. J. J. Atkinson had calculated the relative thickness of cement tubbing required as compared with cast-iron, and proved that in no case could cement resist a head of more than 75 fathoms of water.†

Mr. R. T. MOORE said that the subway tunnels at Glasgow and of the Loch Katrine waterworks were lined with concrete without any brickwork, centres being used in the usual way and the concrete rammed in. The system, he thought, would prove equally efficient for lining shafts.

* *Trans. N. E. Inst. Min. Eng.*, vol. ix., page 73. † *Ibid.*, vol. ix., page 175.

Prof. Arnold Lupton wrote, asking what were the depths of the top and bottom of the tubbing below the surface, what was the depth below the water-level, and how was the cement lining joined to the strata at the top and bottom and at the Cecilia mine, and at the Leopoldshall mine, and what was the thickness of the cement wall at the former mine? Referring to the costs of iron, brick, and cement tubbing, as given in the paper, perhaps Mr. Brough will explain why the cement tubbing was cheaper than brickwork, the cost of cement being, say, £2 per ton, and of bricks of first-class quality, say, 10s. per ton? Was the cement mixed with sand or with gravel to make concrete blocks, and if the latter, are the blocks of the same composition all through, or was there a facing of cement?

Mr. Bennett H. Brough, in reply to the discussion, wrote that he regarded the President's reference to Mr. Watson's paper read in 1861, a paper that had escaped his notice, as a further proof that there was nothing new under the sun. Ample illustrations of the re-invention of forgotten appliances and methods were met with in mining practice. For example, the mine ventilator, called by the Cornish miners the duck machine, and by the Germans the Hartz air-pump, received a premium as a new invention from the Society of Arts, though there was very little reason to doubt that it had been in use for upwards of a hundred years on the Continent. Again, an ingenious co-ordinate protractor for plotting mine surveys was patented in the United States in 1886, although the identical instrument was described and figured two hundered years previously in a German treatise on mine surveying. Numerous other examples might be cited. In the case of the cement blocks, however, the German inventors could lay claim to originality in the shape, which facilitated the setting of the blocks. Mr. Watson's blocks appeared to have been of small size and shaped like ordinary bricks. The objections to the use of cement brought forward during the discussion appeared to be serious ones. It was possible, however, that means might be devised to enable the cement to withstand a greater head of water by enclosing a framework of iron in the block, a method that was largely adopted for fireproof floors. The cost of cement would be less than that of brick tubbing on account of the large size of the blocks, and the facility with which they were placed in position. He would obtain the information asked for by Prof. Lupton, and would, he trusted, be able to lay it before the meeting at the adjourned discussion.

DISCUSSION UPON MR. WILLIAM BOBY'S PAPER ON "THE MELDRUM SELF-CONTAINED DUST-FUEL FURNACE;"* AND MR. BRYAN DONKIN, JUN.'S PAPER ON "STEAM BOILERS WITH FORCED BLAST: THE PERRET SYSTEM FOR BURNING DUST AND REJECTED FUELS: WITH NOTES ON TESTING BOILERS."†

Mr. T. Colquhoun said the system described by Mr. Donkin—the Perret system—had been adopted at the West Stanley collieries some two years ago with the primary object of using up waste fuel, and it had been very successful. In the first instance, they used coal-mud and coke-breeze, the coal-mud being the refuse of the coal-washing apparatus: it was allowed to settle, and the water having been run off it was collected and used in about equal proportion with coke-breeze. They found it to answer very well, and it led them ultimately to try other kinds of waste fuel. They found that an inferior bottom coal would burn very efficiently. The system acted so satisfactorily with one boiler that they ultimately adopted the system for four boilers. During the early months of last year they burnt the bottom coal alone of the Shield Row seam for raising steam. They had used the duff of the upper seams, which was practically unsaleable, and it answered very well. When they had only one boiler on the Perret system, the remainder of boilers being ordinary Lancashire without forced draught, they had tried these different kinds of fuel with the ordinary Lancashire boiler, but failed to get steam for the current requirements of the pit, and they were compelled to burn a fuel which could otherwise have been sold or utilized for making coke. His experiments as regarded evaporative results had been very limited, and he would like to have them completed before placing them before the Institute.

The President hoped the results of the experiments would be communicated to the Institute at a later date.

Mr. Colquhoun said he would be most happy to submit them to the members.

Mr. A. L. Steavenson asked Mr. Colquhoun if it would not have been better to have utilized the coke-oven heat for raising steam?

Mr. T. Colquhoun replied that the waste heat of thirty-two coke-ovens was applied to two boilers for raising steam.

Mr. A. L. Steavenson said he noticed that Mr. Donkin, in his

* *Trans. Fed. Inst.*, vol. iii., page 250. † *Ibid.*, vol. iv., page 154.

paper, took as his unit the quantity of coal required to evaporate 1,000 gallons of water. He thought it would be better to stick to the quantity of water evaporated per lb. of coal, with which they were all familiar. Apart from mechanical questions, of which he had not had any experience, he thought the systems described appeared very satisfactory.

Mr. W. O. Wood said five Perret furnaces were working at Murton colliery. He had not tried any experiments to ascertain the evaporative power of the furnace, but its value consisted in burning refuse from the coal-washer which formerly went over the waste heap, with about as good results as obtained from good fuel consumed in the ordinary way. They got roughly the same amount of power from five boilers fired with refuse as from five fired under ordinary conditions with peas and duff. Possibily in the Meldrum arrangement, in which a steam blower was used, the expenditure of steam might not be greater than it would be in the case of a fan applied to the Perret furnace It was a similar arrangement, and the results would no doubt be about the same, but he had not tried it, and it was only a question of whether steam was more economically used in the engine (driving the fan) or in the blower.

Mr. M. Walton Brown stated that with these furnaces it would be possible to burn household ashes and refuse, whose disposal was frequently a source of cost at collieries. Mr. Donkin stated in his paper that $2\frac{1}{2}$ lbs. of water could be evaporated per lb. of house refuse, and it would probably be found that this efficiency would be exceeded, were ashes from workmen's houses, supplied with free coal, utilized for the purpose of raising steam.

Mr. T. Colquhoun said the waste fuel used was rubbish, which would otherwise have been tipped over the heap; such rubbish raised steam, when burned with forced draught, in such quantities that they had not had to add to their boiler power.

The President said he had been told by a gentleman who had given some attention to the subject, that they could get 50 per cent. more work out of a boiler under ordinary conditions than with forced draught.

Mr. T. Colquhoun said he had found the results were about the same with waste fuel and forced draught, as with good fuel without draught.

Mr. R. Laverick referred to a colliery where the Meldrum furnace was in use, and with factory small coal (peas and duff mixed) they got 17 per cent. more steam than with the ordinary furnace using the same class of coal, the total saving was estimated at 30 per cent.: this was effected by using an inferior class of coal, with less labour in stoking, and with less wear-and-tear owing to the furnace bars being kept cool, which

prevented clinkers setting upon them. There was more perfect combustion, the quantity of ashes being less than one-half what was produced in the ordinary furnace with natural draught.

Mr. C. C. Leach said they would get more effect from the boiler but less from the coal.

Mr. R. T. Moore said that in an instance in which they had adopted forced draught they simply took a 12 inches pipe to the boiler from a forcing fan which ventilated the colliery at about 1 inch gauge. They found less difficulty in keeping up steam; and it enabled them to use a very much thicker fire. With this inch of pressure the air was forced through the bars, the coal in the centre of the fire was wholly consumed and they got just as good results from inferior stuff as they had before from better stuff without forced draught.

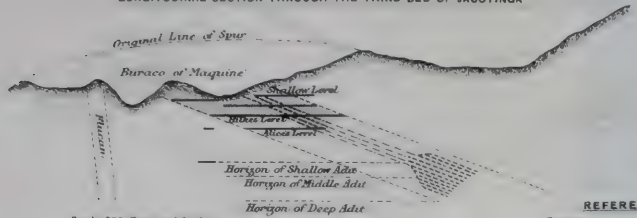
Mr. M. Walton Brown said that Mr. Donkin gave a table of experiments which showed that the evaporation of water increased from 1,277 lbs. per hour without draught to 2,366 lbs. per hour with forced draught, but the efficiency was practically the same; without forced draught 9·9 lbs. of water were evaporated per lb. of fuel, whilst with forced draught it was 10 lbs., a difference of only 1 per cent.

Mr. B. Donkin, Jun., wrote, in reply to the discussion, he was of opinion, as to the unit of comparison for steam boiler results, that the cost of fuel per 1,000 gallons of water evaporated, was decidedly the best. The pounds of water per pound of fuel should, however, always be added, as it gave the efficiency of a given fuel under a given boiler, but did not take into account the cost of evaporating 1,000 gallons of water for any given fuel, or the money paid to evaporate this quantity. This system was now generally used in the London districts to compare the efficiency of different boilers with different fuels. The boiler and apparatus that evaporated 1,000 gallons of water at the lowest cost of fuel was the most economical. Forced draught, now so largely used in boilers, could always be made to give greater evaporative results than with ordinary natural draught, and at lower costs. Combustion was more perfect, as shown by analysis of the waste gases. He might add that some fuels could hardly be burnt unless with forced draught.

The following paper was taken as read :—"The Choice of Coarse and Fine-crushing Machinery and Processes of Ore Treatment—Part II." By Mr. A. G. Charleton.

FIG.1.

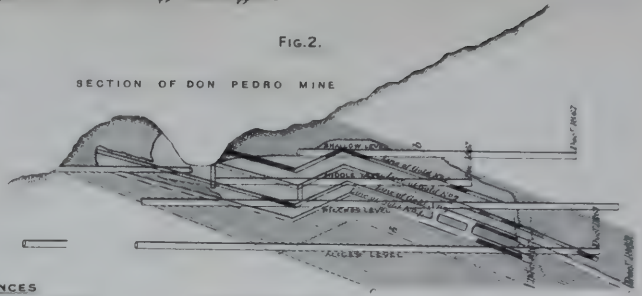
LONGITUDINAL SECTION THROUGH THE THIRD BED OF JACOTINGA



Scale 600 Feet to 1 Inch

FIG.2.

SECTION OF DON PEDRO MINE



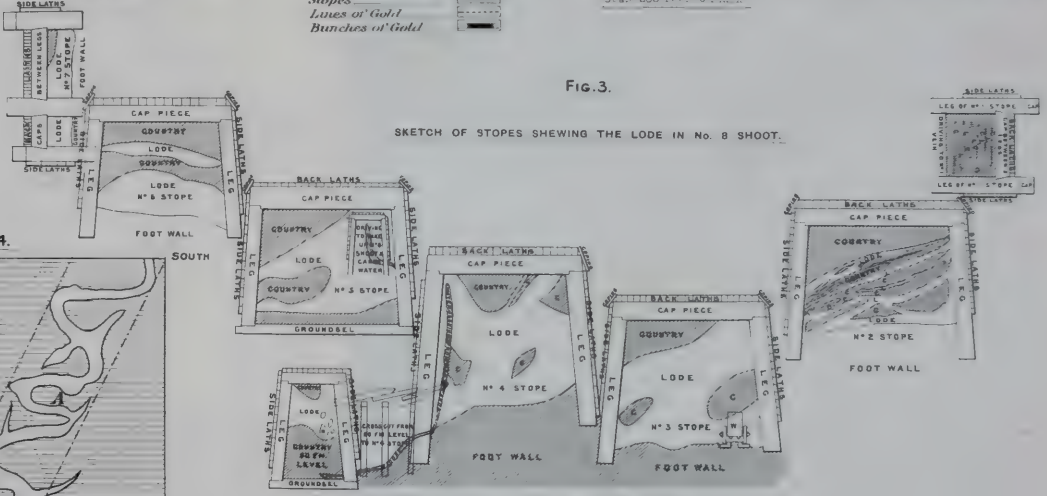
Scale 200 Feet to 1 Inch

REFERENCES

Stopes
Lines of Gold
Bunches of Gold

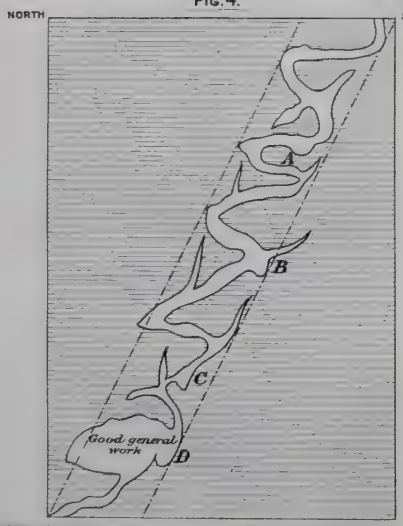
FIG.3.

SKETCH OF STOPES SHEWING THE LODE IN No. 8 SHOOT.



Scale 10 Feet to 1 Inch

FIG.4.

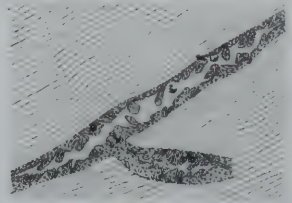


REFERENCES

A First Line of Gold
B Second do
C Third do
D Fourth do

Scale 16 Feet to 1 Inch

FIG.5.



Cross Section of the Lode at the Don Pedro Mine, at a Right Angle with the Dip

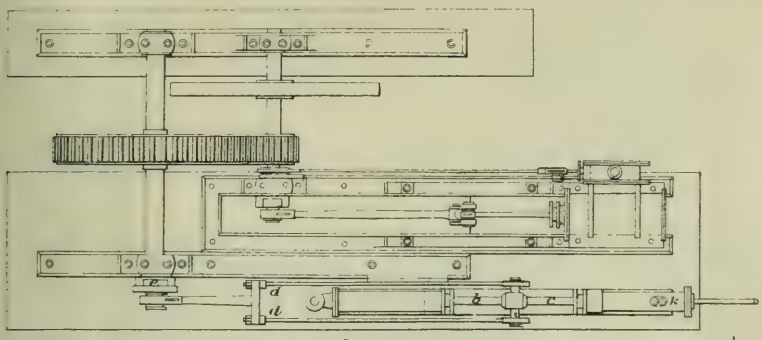
REFERENCES

Dark Auriferous Jacotinga or Lode
Auriferous Earthy Matter
Lines of Gold Nos 1 2 3 4 & 5.

And.w Reid Sons & Co.ths Newcastle

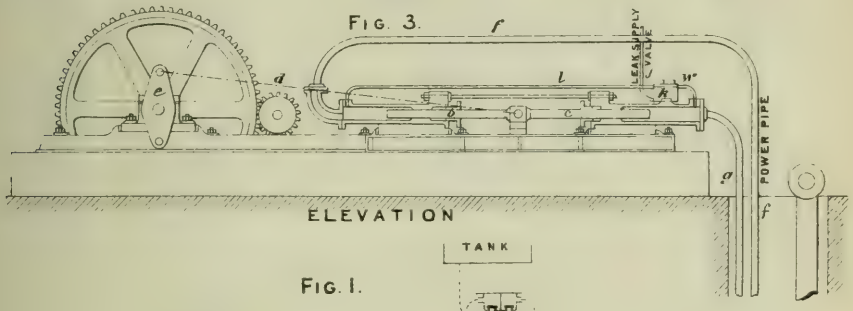
To illustrate Mr R.T. Moore's Paper on "Joseph Moore's Hydraulic Pumping Arrangement."

FIG. 2



PLAN

FIG. 3.



ELEVATION

FIG. 1.

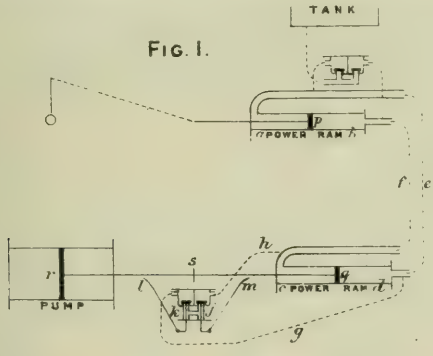
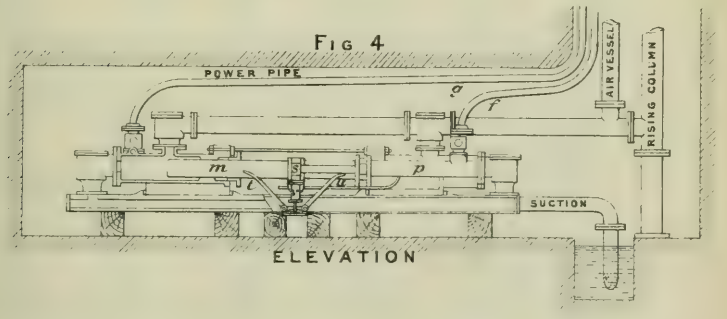
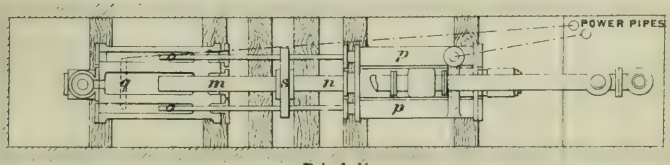


FIG 4



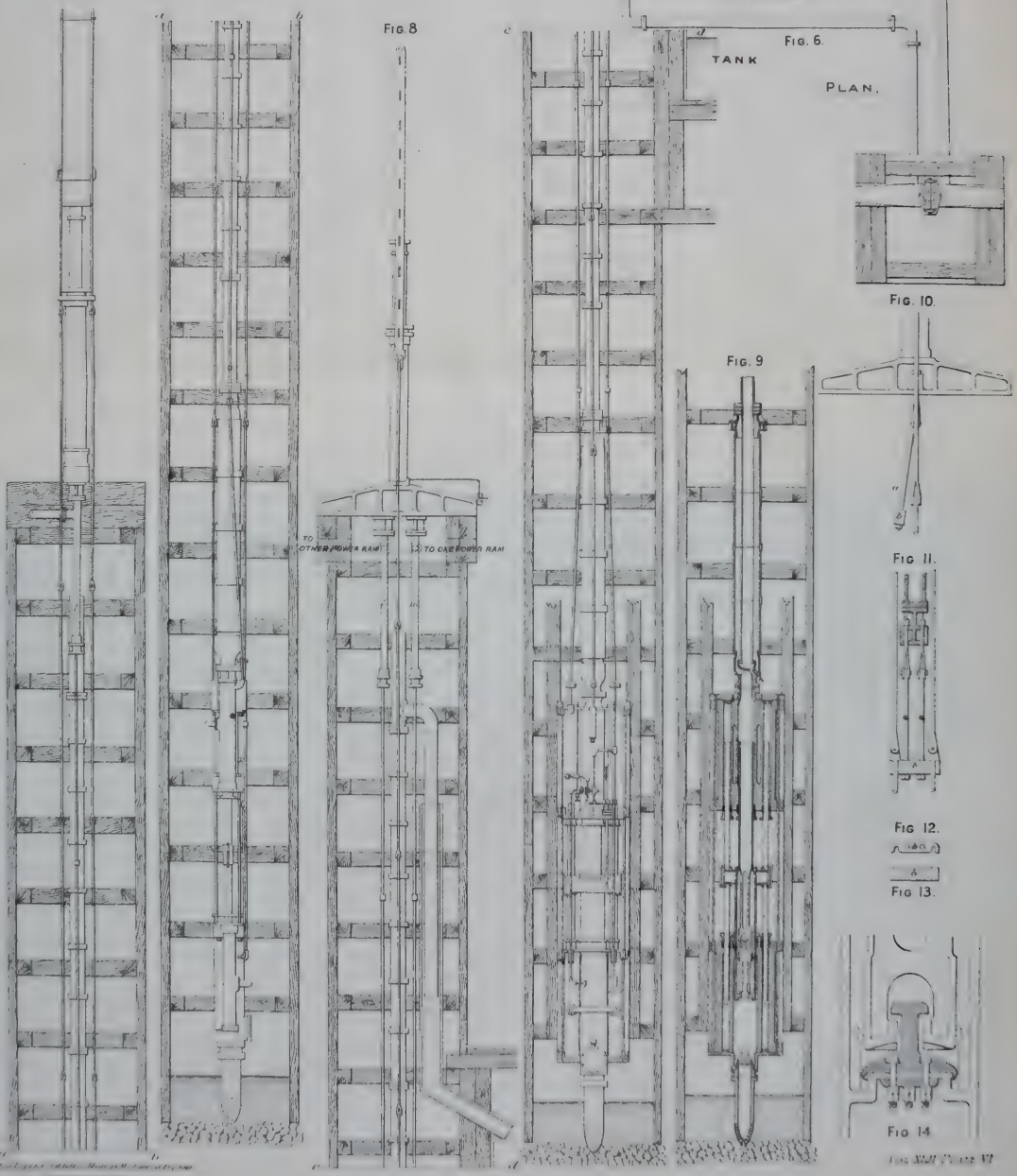
ELEVATION

FIG. 5.



PLAN

To illustrate Mr R.T.Moore's Paper on "Joseph Moore's Hydraulic Pumping Arrangement."



Federated Institution of Mining Engineers
Transactions 1892-93.

To illustrate Mr Bennett H. Brough's "Note on the use of Cement in Shaft Sinking."

Vol. IV. Plate XIX

Fig. 1.

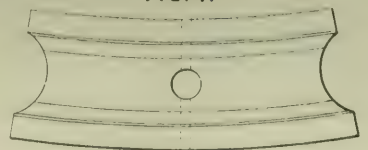


Fig. 2.

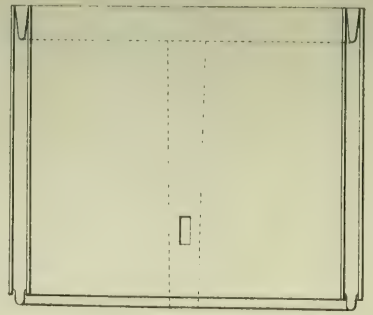


Fig. 3.

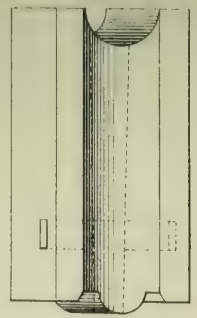


Fig. 5.

Fig. 6.

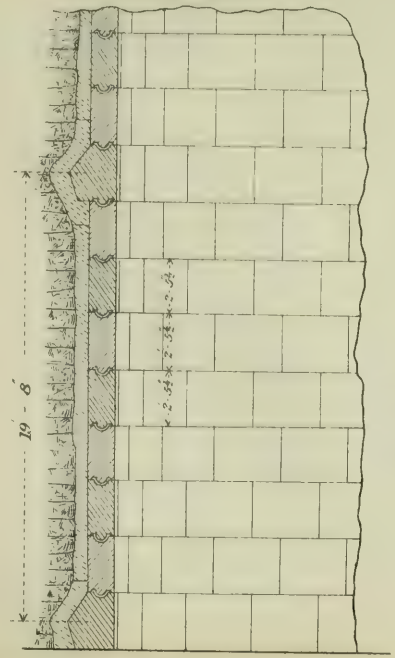


Fig. 7.

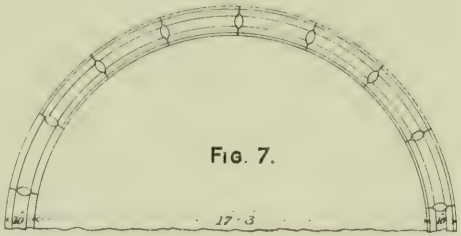


Fig. 8.

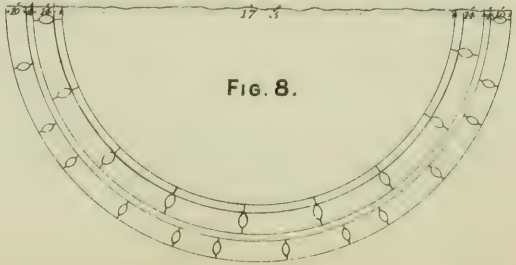
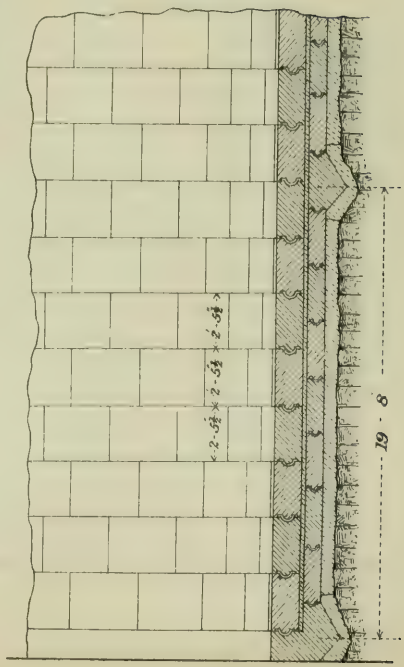


Fig. 4.

Fig. 9.



North of England Institute of Mining & Mechanical Engineers
Transactions 1892-93.

And.w Reid Sons & Co.e Lithos Newcastle

Vol. XLII Plate VII

THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.*

By A. G. CHARLETON.

Part II.—The Relative Scope of Rival Methods of Ore Treatment.

Having examined the metallurgical nature of the various ores and processes (which have been superficially touched upon in the briefest possible manner), we are in a better position to proceed to the still more complicated and difficult questions of the relative prime cost of plant, working costs, and the percentage of gain or loss in rival methods of ore treatment. These are liable to differ in different countries, and under varying conditions even in the same neighbourhood, according to the nature of the ore, cost and quality of labour, material and supplies, as well as the economical or uneconomical manipulative or mechanical modifications in treatment, which can be introduced in each case, on which probably more than anything else, the individual economy or wastefulness (the profit and loss in fact) in works of the same class will depend.

To form an estimate of the prime cost of plant, we must either know the ordinary gross cost of a mill of a certain size and description in any particular locality, or, what is better, the actual quantities and prices of material of different kinds used, and the number of shifts worked by different classes of artizans at specific wages rates, in the construction of the foundations and superstructure of the building, and the erection of the machinery; as well as the price of the machinery itself, the weight of all the transportable material, and freight rates.

The collection of such facts is unfortunately too rare in mining operations to make it always an easy matter to form estimates of the kind applicable to different localities. It is only by piecing together fragments of information obtainable here and there that one can get anywhere near the truth.

Under the category of rival processes, the relative costs and results of treating the following classes of ore may be compared:—

1. Pyritic gold ores by chlorination in vats and chlorination in barrels, and by the new cyanide process, as well as by grinding (without roasting) in pans and arrastras.

* *Trans. Fed. Inst.*, vol. iv., page 233.

2. Silver ores in general,* by wet and dry crushing and pan-amalgamation, the patio and other processes, as compared with the ordinary and Russel lixiviation processes.
3. Free-gold ores by battery-amalgamation, both with and without grinding, and concentration.

The costs and loss in coarse-concentration and smelting have already been considered, and will not be referred to again.

There is no branch of ore treatment which admits of wider differences of internal detail than wet concentration (producing a corresponding effect on its cost†), but such points cannot possibly be entered into here; and as regards dry concentration, it cannot at present seriously be regarded as a rival to wet methods, and may therefore be passed over entirely.

The concentration of sulphide ores of gold and silver often depends on the relative proportion the sulphides bear to the gangue. No sharp line of distinction can be drawn between ores which are suitable for concentration or not, as average value and local cost of treatment enter into the calculation, but roughly speaking, such ores as contain 40 per cent. of sulphides are not usually suitable for concentration. Frequently an ore which will average 20 to 50 per cent. of sulphides, coming from the mine, is best divided by hand selection into first-class ore (to be treated by smelting, or some other process), and a second and poorer class for concentration.

In a mine where the ore runs in seams and pockets of solid mineral, it may happen that 100 tons of crude ore will contain on an average 20 per cent. of sulphides, which can be separated by hand selection into 20 tons of 75 per cent. sulphides and 80 tons of $6\frac{1}{4}$ per cent. sulphides. When this can be done cheaply it is often better than subjecting the whole 100 tons to the costs and losses of concentration, whilst it renders commercially possible a more expensive but more efficient process than the original ore could bear.

A point to be remembered, however, in concentration is, that clean, that is to say pure concentrates, are often as important to obtain as

* The term is applied to ores which do not contain more than, say 10 to 15 per cent. of lead. Ores with more than 15 per cent. of lead are known as silver-lead ores, and are smelted when circumstances admit of it, which is the only admissible proposition for their treatment. Carbonate of lead and galena are converted by roasting sometimes into oxy-chloride and sometimes into sulphate; the former goes into the amalgam, whilst the latter does not amalgamate, and this circumstance explains why some bullion of plumbiferous silver ores is free from lead, whilst other bullion sometimes contains 600 parts of lead in 1,000, according to the condition of the lead in the roasted ore.

† In western mining camps, the cost generally runs between 4s. 2d. and 12s. 6d.

clean tailings (which latter are essential for close saving). You may have the one without the other, and although a clean separation of the metallic components of the ore is almost always advisable, in some instances it may be desirable to leave a certain proportion of sand or gangue in the headings, either as a flux for smelting (as previously pointed out) or to lighten the pan-charges in amalgamation.

Vat and Barrel-chlorination.

For working a few tons of concentrates per day the regular Plattner process, with fixed tanks and long exposure to gas, is as convenient and economical as any of the improved processes, because the plant is simple and cheap to erect, and very few hands are required to run it.*

The production of gas is a simple question of relative costs of different chemicals delivered at the mine. For working large quantities of ore daily, especially if of comparatively low grade, which requires economy in crushing, roasting, gas consumption, labour, and time, the use of barrels, in place of fixed tanks, effects a saving in time and labour, and makes special watchful skill less important in the whole process than in the Plattner method, so that one of the modifications of the barrel system would be naturally adopted on crude ore and in dealing with large quantities of concentrates.

The attrition of the ore particles and the thorough turning over of the charge in the presence of nascent chlorine in the barrels, has also some effect upon the results, especially if there be coarse-gold present which has to be amalgamated afterwards, as the gold, it is said, is attacked with much greater avidity by the mercury in consequence of the clean surface created by the action of the chlorine and attrition combined. It is necessary, however, that the tailings should first be washed, by continuing the leaching, till all trace of chlorine has disappeared, and that they should be treated by amalgamation at once. If left to dry, the gold particles assume a red colour and amalgamate with great difficulty.

The most successful furnaces in general use for chlorination-roasting appear to be of three types:—

1. Plain or step-hearth reverberatories furnaces, of the Fortschaufelung class, with a hearth surface of about 12 feet wide by 75 feet long.
2. Horizontal round-hearthed mechanical furnaces, of the English Brunton-calciner type, about 12 feet in diameter.
3. A modified form of the Spence furnace.

* *Gold Amalgamation and Concentration*, page 28.

To which may perhaps be added an altogether new modification of the Fortschaufelung furnace, which will be alluded to later on.

When battery-amalgamation and concentration precedes chlorination, Mr. A. Thies* recommends the use of brass-wire screens in place of slot-punched sheet-iron ones, stating that when using 36-mesh brass-wire in place of 40-mesh slotted-iron, at Haile, in North Carolina, he obtained a far more uniform pulp for concentration, whilst the average life of the wire was found to be six weeks as compared with the sheets, which had to be thrown out in fourteen days, a matter which is worth noting. German practice certainly supports this view so far as concerns the effect of wire screens on the concentration results.

Dr. Egleston states that the correct amount of charge, in roasting gold-sulphides in the reverberatory furnace, is usually 10 to 12 lbs. per square foot of hearth; an ordinary charge amounting to about a ton at a time on each division of the hearth, of which there are usually three.

A furnace of this description, designed to handle from 3 to 4½ tons of concentrates per 24 hours, 75 feet long by 11 feet wide inside, requires for its construction 40,000 common bricks, 8,000 firebricks, and 5 barrels of fireclay, and the ironwork, tie-bars, etc., exclusive of T-rails, weighs 11,000 lbs. A furnace with a 14 feet by 60 feet hearth requires 86,000 red bricks and 15,000 firebricks, and costs about £626 in Pueblo or Denver.

In vat-chlorination, the rest of the plant to correspond, covered in (as the furnace must also be), consists of a drier, a set of fixed or rotating leaching-vats, with (in some cases) silver-leaching tubs added, precipitating-tanks, and reagent tanks, well and pump, lead-generator, heating-pan and wash-bottle, a drying muffle and melting furnace, and a press and filters for precipitate, as well as filter-tubs for the base metals† in the waste-solutions, and cars for handling the ore, overhead tackle for lifting the vat-covers and tools, etc.

The cost of a plant of the kind in California, with a capacity of 6 tons in 24 hours, is stated‡ to vary between 6,000 and 7,000 dols. (£1,250 to £1,458 6s. 8d.). A small plant would cost about half the former sum.

The Thies modification§ of the Mears process (one of the most successful unpatented systems of barrel-treatment), requires the sub-

* "The Thies Process of Treating Low-grade Auriferous Sulphides."—*Trans. Am. Inst. Min. Eng.*, vol. xix., page 601.

† If present and worth the extra treatment.

‡ "The Milling of Gold Ores in California," by John Hays Hammond. *Eighth Annual Report of the California State Mineralogist*, page 696.

§ A description of this process is given by Dr. Egleston in *The Metallurgy of Gold and Mercury*, page 664, *et seq.*

stitution of rotating barrels for vats, the chlorine being generated inside the barrels, by the action of sulphuric acid (66 degs. B.) on bleaching powder, which is added to the charge of water and ore under ordinary circumstances in the proportion of 10 to 30 lbs. of chloride of lime to about 15 to 30 lbs. of acid per ton of charge of roasted ore in each barrel.

If copper is present more of these reagents are required.

The barrels must be run by steam or water-power, the latter being, of course, the most economical, when available.

The rest of the plant, though differently arranged, is substantially the same as that for vat-chlorination, and will cost not less than £620 for works with a capacity of 5 tons per day; the weight of the machinery will be about 17 tons.

The cost of the vat process at the works of the Plymouth Consolidated Company,* where 100 tons of concentrates is treated per month of thirty days (leaching taking twenty-four days), is given as follows :—

	£	s.	d.	£	s.	d.
Roasting—						
3 men at 10s. 5d. per day for 30 days	46	17	6			
$1\frac{3}{4}$ cords of wood at 17s. $8\frac{1}{2}$d.	46	9	$8\frac{1}{2}$			
54 lbs. of salt at 3s. 8d.	2	10	$7\frac{1}{2}$			
				95	17	10
Chlorine (employing two generators)—						
60 lbs. of manganese per day at £9 15s. 10d. per ton	7	1	0			
68 lbs. of salt „ 3 2s. 6d. „	2	11	0			
120 lbs of acid „ 12 10s. 0d. „	18	0	0			
				27	12	0
Leaching—						
40 lbs. of sulphuric acid for settling-tanks for 24 days	6	0	0			
40 lbs. of sulphuric acid for making sulphate of iron for 24 days	6	0	0			
Wages of leachers (2 men at £1 2s. 11d.†) for 30 days	34	7	6			
Wages of foreman	26	0	10			
				72	8	4
Total cost per month ...				£195	18	2

or £1 19s. 2d. per ton.‡

* *Trans. Am. Inst. Min. Eng.*, vol. xv., page 305.

† Two men on day shift attend to all the work of handling the ore after it is leached. The head man receives 12s. 6d., the others 10s. 5d. Only three tankfuls are leached every four days; but the men are employed steadily. The sulphate of iron is manufactured on the spot.

‡ To this should be added for assays, repairs, supplies, insurance, taxes, water, interest on invested capital, and deterioration of plant, as well as proportion of general expenses and superintendence, about 16s. 8d. additional per ton, making the total cost of working £2 15s. 10d. per ton.—*Eighth Annual Report of the California State Mineralogist*, page 48.

At the Providence Works* two furnaces are used, which have a capacity of 9 tons in 24 hours. Each furnace consumes 1 cord of wood, and the cost per day can be reckoned as shown in detail on page 166. This makes the cost of treatment per ton of sulphides to be 14s. 9½d when the works are run at full capacity.

As the ore, however, only contains about 7 per cent. of sulphides, or 4⅓ tons in the 62 tons milled daily, this quantity does not keep the two furnaces running full time, though both of them are in constant operation. As most of the expenses remain the same, whether running at full capacity or not, the actual cost of chlorination, figured on a working basis of 4½ tons treated daily, approximates, therefore, more nearly to £5 13s. 10d. per day, or £1 5s. 3½d. per ton of concentrates, as detailed below. Add to the above sum (£5 13s. 10d.) the cost of milling per day, viz., £11 19s. 7d., and it will be found to represent a total outlay of £17 13s. 5d. per day, or a total charge of 5s. 8½d.† per ton of crude ore raised from the mine.

This estimate makes no allowance for general supervision, interest on first cost, or deterioration. The conditions of treatment in these works are, however, very special, and can hardly be considered as a basis to estimate upon in most instances.

The estimated cost of the treatment of 4½ tons per day is:—

	£	s.	d.
Labour	2	13	1½
2 cords of wood at £1 0s. 10d.	2	1	8
14 lbs. of manganese at 1⅜d.	0	1	7
126 lbs. of salt at ¼d.	0	5	3
104 lbs. of acid at 1d.	0	8	8
Lime, sulphur, and calcium hyposulphite ...	0	0	7½
Illuminating	0	0	10
Extras	0	2	1
Total cost per day ...	£5	13	10

or £1 5s. 3½d. per ton.

At the Alaska Treadwell Company's works the cost for the year ending May 31st, 1891, was as follows, calculated on a basis of 3,568 tons of concentrates treated:—

* *School of Mines Quarterly*, vol. v., and Egleston's *Gold*, page 657.

† 1s. 10½d. of this is chargeable to the chlorination.

Labour—	Cost per Ton. (American Money.) Dollars.	Cost per Ton. (English Money.) £ s. d.	Gross Cost. Dollars.	Gross Cost. £ s. d.
Foremen	·5600	...	3,006·43	...
Engineers and foremen	·0534	...	286·38	...
Gas generator	·2089	...	1,121·29	...
Salters	·0855	...	458·24	...
Floormen	·7604	...	4,080·83	...
Roasters	2·9400	...	15,782·49	...
Carmen	·2077	...	1,114·72	...
Carpenters	·0039	...	21·39	...
Labourers (white) ...	·0999	...	536·62	...
„ (Indian) ...	·0309	...	166·00	...
Teamster	·0806	...	433·11	...
	$5·0312	£1 0 11½	$27,007·50	£5,626 11 3
Supplies—				
Acid	1·0220	...	5,486·38	...
Wood	1·5230	...	8,175·93	...
Coal	·0543	...	284·80	...
Furnace supplies ...	·0270	...	145·19	...
Manganese and salt ...	·8095	...	4,345·86	...
Pipe	·0089	...	47·90	...
Generator fittings ...	·0038	...	25·93	...
Leaching tanks... ...	·0638	...	339·05	...
Miscellaneous	·1353	...	727·26	...
Rake-heads	·0360	...	193·50	...
Car wheels	·0037	...	27·50	...
Repair account... ...	·1352	...	720·08	...
Electric light account...	·1077	...	578·49	...
Haulage account ...	·0549	...	294·77	...
	$3·9852	£0 16 7¼	$21,392·63	£4,456 15 11½
Total cost per ton of sulphides ...	$9·0164*	£1 17 6¾	$48,400·13	£10,083 7 2½

Total gross cost.

* In the six months ending November 31st, 1892, this charge was reduced to $8·42 = £1 15s. 1d. per ton of sulphides. As an instance of economical management, the total costs of mining and milling per crude ton extracted may be cited as follows, the ore being quarried in open benches :—

	Cents.		s.	d.
Mining	65	=	2	8½
Milling and concentrating	33	=	1	4½
Chlorination	19	=	0	9¼
General expenses (mine)	8	=	0	4
General expenses (in San Francisco)...	2	=	0	1
Bullion freight, etc.	5	=	0	2½
Total cost ...	$1·32	=	5	6

per crude ton of ore, but the average for some time previous was 6s. 3d. per ton.

The roasting at these works is done in Spence automatic furnaces, each with four hearths, which roast 8 tons a day. On the hearth next the lower one, 3 per cent. of salt is added with a special spoon. The ore, instead of being as formerly stirred by fixed rakes, is now rabbled by oscillating ones, which tip every three minutes. On the upper drying-hearth they last a long time, but on the lower ones, where the heat must be great to drive off the last traces of sulphur, their life is only about three months.

Formerly the rakes stopped between the forward and backward motion just over the flue, now they are made to go beyond it, and are no longer exposed, as they were at this point, to the sulphurous gases and hot falling ore, which clogged the end of the shelves and gave such trouble at the Haile mine, finally causing the abandonment of the Spence furnace there for roasting fine ores, although admittedly a most excellent mechanical roaster for coarser grades of material in its original shape.

With the important modification above alluded to it seems doubtful (judged by the above facts) how far Mr. Thies' condemnation of the Spence furnace* (in works with a capacity of 3 to 4 tons) for fine ores (where a dead sweet roast is required) is entirely justified.

The labour involved in vat-chlorination on an average may be taken at :—

1 man, who works 10 hours, to bring in wood and ore.
1 man at the furnace, who works 8 hours.
1 chlorinator, „ 8 „
3 labourers (1 on shift by day, and 2 by night).

At Sutter Creek it takes 8 men for the whole 24 hours, one chlorinator, one helper, five men at the furnace, and one who wheels ore.

The consumption of chemicals per ton of ore is about as follows :—

Salt in the furnace	40 lbs.
„ „ generator	6 „
Manganese „	6 „
Sulphuric acid generator (66 degs. B.) ...	20 „

The stock of chemicals ordinarily kept on hand, on the scale of operations of the Alaska Treadwell Company, seems to be :—

	£	s.	d.		£	s.	d.
Salt, about 53 tons, valued at	2	17	11	...	153	9	7
Manganese, 13 „	9	15	10	...	127	5	10
Sulphuric acid, 35 tanks, at	8	18	10½	...	313	0	7½
Total value				...	£593	16	0½

* *Trans. Am. Inst. Min. Eng.*, vol. xix., page 610.

The works in Grass Valley, where vat-chlorination is most extensively used, guarantee an extraction of 90 per cent. of the gold, and 60 per cent. of the silver, but the saving more often amounts to between 90 and 94 per cent. of the gold in pyritic-concentrates, and over 60 per cent. of the silver, if the tailings are leached to obtain it.

The ore to be properly roasted ready for leaching should maintain a nearly vertical face when made into heaps on the finishing hearth, and cut down with a spadelle. It should show no bright specks, but be inclined to become black, which will generally be in 7 or 8 hours. The largest amount of gold has been shown to be in the best condition to be leached and to consume the least amount of chlorine in chlorinating it, when the ore falling in the furnace, in turning it over, has a slight violet colour. When the ore sparkles, and the sparks are numerous and bright, it shows that the roasting is not properly finished, and more salt has to be used.*

A ton of roasted ore will occupy usually $24\frac{1}{2}$ cubic feet. This quantity is derived from 2,800 lbs. of raw sulphides, which occupy about $13\frac{2}{3}$ cubic feet per ton (2,000 lbs.). A ton of sulphides will weigh from 1,450 to 1,700 lbs. after roasting, and occupy about $17\frac{1}{2}$ cubic feet. The roasted ore ought not to contain more than $1\frac{3}{8}$ per cent. of sulphur.†

At the Plymouth Consolidated works, the Fortschaufelungs-ofen used for roasting is 12 feet wide by 80 feet long, including the fire-box, the hearth being a continuous plane, but the charges, of which there are three in the furnace at one time, are kept entirely separate. They are called by the furnacemen the drying, burning, and cooking-compartments. In the middle division the ore is spread out very thin, and occupies about double the space of either of the others.

The furnace is worked by 8 hours' shifts, a charge being drawn and added in each shift. The charges weigh 2,400 lbs. and carry about 10 per cent. moisture.

The ore averages about 20 per cent. of sulphur, and just before the sulphur ceases flaming in the second division of the furnace 18 lbs. or $\frac{3}{4}$ per cent. of salt is added to the charge. Care must be taken to keep the concentrates damp until they are introduced into the furnace, or a decomposition of the pyrites begins, forming lumps which do not roast, and

* The heat on the finishing hearth must be maintained at a lively bright red, but not at a white heat, else the gold particles would melt, which, with a good magnifying glass, can be easily detected; after washing off the iron the gold appears then in minute globules, the chlorination of which is more difficult.

† Egleston, *Gold*, page 622.

consequently cause a loss of gold in the residues from leaching. The roasted ore from each shift is kept by itself on the cooling-floor until a tankful (about 4 tons) has accumulated from a single man's shift, and it is then worked by itself. This enables the foreman to better control the work of roasting, for if one lot out of three works badly, it points to the fault being with the furnaceman; whereas, if all three give unsatisfactory results, it may be presumed to be owing to a change in the ore, and the roasting must be modified.

The vats* for chloridizing the roasted ore are 9 feet in diameter by 3 feet in height; they are four in number, and are slightly inclined forwards to drain them completely. The bottom of each tank is composed of a filter about 6 inches thick, consisting of light strips of $\frac{3}{4}$ inch wood laid on the bottom at intervals of about a foot. Across these are placed 6 inch boards, spaced an inch apart. On this loose floor coarse lumps of quartz are spread, and on the top of this again quartz-sand, until a depth of 6 inches is obtained. Finally, this sand filter is covered over with another loose floor, the boards lying cross-wise to the loose floor beneath and pretty close together. This upper floor is merely to facilitate shovelling the charge out when it has been gassed to permit the removal of the leached ore without disturbing the filter.

The ore to be chloridized must be damp (about 6 per cent. moisture). The working test is to take a handful of ore and squeeze it, then open the hand, and if the lump immediately begins to crumble and fall apart (not run) the ore has the requisite amount of moisture. The damp ore is screened into the tank so as to lie as loose as possible and thus facilitate the penetration of the chlorine. A coarse screen of one-and-a-half mesh is used for this purpose.

The tanks are only filled up to within about 3 inches of the top, (to ensure the entire contents being covered with water in the subsequent leaching,) otherwise there would be great difficulty in washing out the soluble gold. As soon as they are full they are gassed. The gas is introduced into the bottom from two opposite sides, and is continued until ammonia held over the ore gives off dense fumes of ammonium chloride, which usually happens in about 4 hours. When this point is reached, covers are placed on the tanks and luted on.†

* The vats should be coated on the inside with asphaltum varnish or a mixture of pitch and tar applied hot, but the former is preferable, as it penetrates better into the pores of the wood.

† It is usual to provide the covers of the tanks with two pieces of $1\frac{1}{4}$ inches gas-pipe 6 inches long, and a square hole, 6 inches by 6 inches, closed with a wooden cover. These pipes are closed with balls of clay during the impregnation of the ore

The two gas-generators* which are employed to charge a tank are allowed to work on till nearly exhausted, when they are disconnected, and the holes in the tank are plugged up. The tanks are usually charged in the morning, and left standing two days. On the third day the ore is leached. The tank is first filled with water, and allowed to stand a few minutes to permit the water to penetrate the ore. If no more water is absorbed, the liquor is drawn off at the bottom, care being taken to keep the tank full of water during this part of the operation, which lasts 4 to 5 hours.

For charging the tank a gunny sack is laid on the ore and held down with a couple of bricks where the wash-water is afterwards to be introduced, in order to better distribute the water in the tank, and prevent it washing out holes and packing the ore.

The liquor from the leaching-vats is conducted to settling or storage-tanks, where about 40 lbs. of sulphuric acid is added (66 degrees B.) and it is allowed to stand 2 to 24 hours. It is then run into the precipitating-vats, where the gold is precipitated with sulphate of iron; the iron solution being added until after stirring, a further addition produces no purple colour. After the gold is precipitated it is allowed to stand from two to three days to settle, when the supernatant liquor is drawn off with syphons into a second settling-tank, where any gold drawn off by the syphons has a second opportunity of settling.

The liquor stands in this tank till it is necessary to run it off to make room for a fresh charge. Very little gold is found in this tank, so it is only cleaned out about once a year.

In the meantime fresh liquor has been run into the precipitating-tanks on the gold already precipitated there. In this way the gold is accumulated till the semi-monthly clean-up, the precipitating-tanks being kept locked and covered. In making a clean-up, the supernatant liquor is syphoned off, the gold gathered up, and placed in a filter of punched iron, lined with filter-paper, and washed with water till all the acid and iron salts are removed. It is then dried, melted in crucibles, and cast into bars.

Under unfavourable conditions the cost of the chlorination process may run up as high as £4 3s. 4d. per ton, but in California it may

with gas. After gassing, the clay is removed, and one of the pipes is coupled to the hose of the water-tank, whilst the other is connected either with another tank ready for chloridizing, or the ashpit of the roasting furnace, partly to utilize the surplus chlorine, as well as to protect the workmen from its injurious effects.

* When practicable, it is a good plan to heat the generators by steam in place of direct heat.

generally be taken as costing £1 17s. 6d. to £3 1s. 6d. per ton, £2 14s. 2d. being in all probability a fair average on a basis of treating 3 tons or more daily.

Mr. G. F. Deetken is responsible for the statement* that with favourable conditions of cheap wood and freight rates, not counting interest, insurance and amortization of a capital of 3,000 dollars, the working expenses in some parts of California do not exceed £1 0s. 10d., and are sometimes as low as 16s. 8d. per ton.

In locating works of this kind it is important to secure a good fall, and they should be terraced and so placed with reference to the prevailing winds that noxious fumes will not be carried in the direction of valuable land or house property.

There should also be a supply of water at hand, delivering 40 to 60 gallons per hour.

Barrel-chlorination.

Mr. Thies, in a letter to Mr. C. N. Aaron (a well-known Californian authority), gives the cost of the Thies process at the Haile mine, North California, as follows :—

Using a double reverberatory furnace, which furnishes 2 tons of roasted ore every 24 hours, with an average consumption of 1 cord of wood, at 5s. 2½d. per cord, and employing four labourers, the cost of roasting the ore amounts to 10s. 11¼d. per ton. Two men can easily treat 4 tons in 10 hours, elevate the ore, and clean out the filter-tanks, of which there are four to each barrel. Arranged on this basis, the cost of roasting and chlorinating amounts to 19s. 3¼d. per ton, as tabulated below :—

	Cost per Ton. s.	d.	s.	d.
Roasting 2 tons of ore—				
4 labourers at 4s. 2d.	8	4		
1 cord of wood at 5s. 2½d.	2	7¼		
			10	11¼
Chlorinating 4 tons of ore—				
2 labourers at 3s. 9d.	1	10½		
1 chlorinator at 8s. 4d....	2	1		
40 lbs. of bleaching powder at 1½d.	1	3		
60 lbs. sulphuric acid at 1d.	1	3		
Sulphuric acid, for making ferrous sulphate	0	6¼		
Repairs, wear and tear...	0	10		
Power	0	6¼		
			8	4
Total cost per ton			19	3¼

* *Eng. and Min. Jour.*, New York, vol. lv., page 244.

We thus have the sum of 19s. 3¼d. for roasting and chlorinating 1 ton of roasted ore, representing 1⅓ tons of raw iron pyrites.

Inside of 7 hours from the time the ore is in the chlorinator, the solutions are ready for precipitation and the tailings are clean.

At the Phœnix mine, in North Carolina, the cost for roasting* and chlorinating by the Thies process is estimated at :—†

	£	s.	d.
Roasting	0	9	1
Chlorinating	0	11	1
	£1	0	2

2,353 tons of concentrates are said to have been successfully treated by this process at the Haile mine and 5,000 tons at the Phœnix mine between May, 1888, and September, 1890.

At the Phœnix mine, a 12 feet revolving pan-furnace is used, which roasts 1 ton of raw ore in 12 hours, with a consumption of ⅜ths of a cord of wood, and 3s. 9d. worth of labour. The cost of power does not exceed 1s. 0½d. per ton.

At the Bunker Hill works, the concentrates are roasted with 1 per cent. of salt, in a reverberatory furnace 40 feet by 12 feet on the outside, with walls 18 inches thick, the stationary hearth of which (7 feet by 18 feet by 2 feet in height, with a working door on each side) terminates in a horizontal revolving hearth, 12 feet in diameter, set at a lower level (giving a fall of 6 or 7 feet). The discharge hole or cub is in the centre of this latter hearth, from which the ore drops into cars.

At the Deloro mine,‡ Canada, where the Mears process was formerly in operation, the ore was roasted (after drying in a revolving drier) in a pair of cylinders,§ one 30 feet in length by 5 feet in diameter, the other 20 feet by 4 feet, set tandem, jacketed with mineral wool and then paper. It is claimed that 10 tons of concentrates have been roasted in these cylinders in 24 hours, so that the extraction of gold has reached 93 to 98 per cent.

The tanks used in the process should be built of wood which has been soaked in linseed oil, dried, and painted with three good coats of white lead or tar.

* Removing the sulphur to within 0·25 per cent.

† Phillips, *Trans. Am. Inst. Min. Eng.*, vol. xvii., page 321.

‡ R. P. Rothwell, *Trans. Am. Inst. Min. Eng.*, vol. xi., page 191.

§ Different furnaces require, on the average, it is said, the following weights of wood to roast a ton of ore (the weights being calculated on the basis of 2,200 lbs. per cord as the average) :—White furnace, 300 lbs.; Bruckner, 900 lbs.; reverberatory, 600 lbs.; Stetefeldt, 200 lbs.—(Report, *Tenth Census, United States.*)

Mr. Thies states that the cost of barrel-treatment depends chiefly on the number of tons chlorinated per day.

The wear on the inner lead-linings of the chlorinators is imperceptible; a chlorinator in use at the Phœnix mine for over five years does not show any wear. The lining is fastened on with bolts from the outside, as it has been found very difficult to burn on lead, when chlorine has acted on it, in making repairs.

At the Haile mine, the fire-assay and value of the ore delivered to the stamps is 4·50 dollars (18s. 9d.) per ton.

The mint returns of bullion gave 16s. 3d. per ton of ore treated, of which 6s. 0½d. is to be credited to the battery, *i.e.*, to free-gold; whilst 10s. 2½d. was obtained from the sulphides.

Taking the assay-value of the ore at 18s 9d., and the actual yield in bullion at 16s. 3d., there is an indicated loss of 2s. 6d. per ton, or 13⅓ per cent. Taking the yield in free-gold at 6s. 0½d. per ton from an ore worth 18s. 9d. per ton (or approximately 32 per cent. of the gross value by assay), we have 68 per cent. to be debited to the concentrates. But the total yield in free-gold, and in gold from the sulphides being 16s. 3d., the ratio of the free-gold saved, to the total amount saved, is approximately 38 per cent., and the combined gold 62 per cent., or a saving of about one-third free and two-thirds combined gold.

The term, combined gold, has been used to express the condition of the gold which is not free. Whether the gold that is not free is chemically or mechanically diffused in the sulphides or both is uncertain, in most cases.

Mr. T. W. T. Atherton* claims to have found gold as a natural sulphide in the pyrites of the Deep Creek mines of New South Wales. He gives an analysis of the ore and the method pursued in experimenting on it, from which his conclusions were drawn.

From 80 tons of ore stamped per day of 24 hours at Haile, 7½ tons of concentrates are obtained from 16 Embrey end-shake tables, which gives as the yield of each concentrator a little less than half a ton.

The average assay-value of the raw concentrates for the 12 months preceding the date of Mr. Thies' paper was £6 5s. per ton, and the percentage of sulphur they contained varied between 40 and 45 per cent. In roasting, this was brought down to 0·25 to 0·40 per cent., whilst the value of the material naturally increased; the raw concentrates worth £6 5s. becoming worth when roasted £8 6s. 8d., or about one-third more per ton.

* *Eng. and Min. Jour.*, New York, vol. lii., page 698.

The assay-value of the tailings thrown away, after chlorinating the roasted ore, was, on the average, only 8s. 4d. per ton, representing an extraction of 95 per cent.

The process had been in successful operation $2\frac{1}{2}$ years, at Haile, when Mr. Thies' paper was written in 1890, and 36,000 tons of crude ore had been treated profitably during that time, prior to which nearly every process had been tried on the ore and failed.

The advantages of the process are:—The small amount of space the plant occupies, speed of operation, high percentage of yield, facility of ascertaining the condition of the charge at any time,* and very slight wear and tear. The only offsets against these advantages are the care and intelligence required to control it, and the need for a small amount of power.

250 to 300 gallons of wash-water are needed per ton of ore treated (an ordinary charge), the water being introduced into the barrel first.

It takes 2 to 4 hours to chlorinate each charge, the barrel revolving at 12 revolutions per minute.

The lead-lining is $\frac{1}{4}$ inch thick, and weighs 12 lbs. per square foot.

The filters must be flooded from below with 4 or 5 inches of water to prevent the ore packing as it falls into them.

The charge must be allowed to drain, and then washed with water as rapidly as possible till the wash-water shows no gold, though still carrying traces of chlorine.

If lime is present, settling-tanks, called stock tanks, are required to settle the liquors which are drawn out of them into the precipitating tanks after standing for 14 to 16 hours.

The precipitating-tanks are large enough to hold the liquors from 3 tons of washed ore. The ferrous sulphate is syphoned into them. The solution must have a decidedly acid re-action in order to be certain that all the lime has been converted into sulphate.

The Providence works and those of Bunker's Hill may be cited, the former as showing exceptionally low cost by the vat process, working 9 tons per diem; the latter unusually high cost by barrel-treatment, treating 2 tons in 24 hours. To compensate for this the loss in the tailings, which used to run as high as £1 9s. 2d. when worked by the ordinary vat process, has been reduced 50 per cent.

* For this purpose a lead valve is arranged in the barrel, so that not only the pressure of an excess of chlorine gas but its actual presence can be ascertained at any time. It does not do to trust to the pressure test alone, other gases being some times given off.

The cost at the Providence works :—

	£	s.	d.
1 foreman	0	12	6
1 white labourer	0	9	4½
5 Chinamen at 6s. 3d.	1	11	3
2 cords of wood at £1 0s. 10d	2	1	8
29 lbs. of dioxide of manganese at 1⅜d.	0	3	4
260 lbs. of salt at ½d.	0	10	10
216 lbs. of sulphuric acid at 1d.	0	18	0
Lime, sulphur, and calcium hyposulphite	0	1	3
Illuminating	0	0	10
Extras	0	4	2
	£6	13	2½

Per ton ... ... 14s. 9½d.

The cost at the Bunker Hill works :—

	s.	d.	£	s.	d.
Roasting—					
2 roasters at 13s. 6½d.	13	6½			
⅝ths cord of wood at £1 5s.	15	7½			
			1	9	2
Chlorinating—					
1 chlorinator at 12s. 6d.	6	3			
30 lbs. bleaching paper at 2d.	5	0			
36 lbs. of salt at ⅜d.	0	7½			
20 lbs. sulphuric acid, 66 B. at 1¾d.	5	3			
Water-power	2	1			
General expenses and loss	12	6			
			1	11	8½
Total cost of 1 ton of ore roasted and chlorinated ...			£3	0	10½

Under efficient management and favourable conditions the cost of barrel chlorination will usually be found to vary between 12s. 6d. and £1 0s. 10d. per ton in America.

The results of treatment by another vat-process, the Pollock patent, are given in the *London Mining Journal*,* treating various ores as follows :—

	Ozs.	Dwts.	Grs.	Percentage Extracted.
Sheba mine tailings...	2	7	21	96
Mixed lot of ore, Transvaal	1	3	22	97
Day Dawn P.C. quartz, Q. 6	6	9	23	97
„ concentrates, Q. 1	1	2	6	97
Swaziland quartz, South Africa	0	16	5	95
City and Suburban quartz, Transvaal	12	10	10	95
Crown ore, New Zealand...	3	16	0	94
Transvaal Gold Extracting Co.	1	10	11	93
„ „ tailings	0	7	8	85
Mount Shamrock tailings	0	8	17	96

* *The Queensland Mining Journal*, Nov. 5th, 1888.

It is stated that the cost of the plant for treating 100 tons a day is about £10,000.

It is certain that ores like those of Meadow Lake district, California, composed of a species of siliceous hæmatite (which in depth will probably be found in an unoxidized condition, consisting mainly of pyrite) will not amalgamate properly, and attempts have been made with various devices and countless processes to work them, but so far without success.

They are said to average £1 9s. 2d. to £2 1s. 8d. per ton or over, and to bear a resemblance to the ores of Bald Mountain district, South Dakota. It is possible that if a sufficient quantity of concentrates could be secured to keep works of the kind running many of these ores could be advantageously handled by the barrel-chlorination process, without a previous roasting, if treated with an oxidizing agent such as nitre cake, and in working the surface-ores even concentration might at first be dispensed with.

An interesting modification of the barrel process (described by John E. Rothwell, *Eleventh Census*, United States, 1890) is to make the chlorination-barrel also the washing and leaching-vessel. This is effected by fixing a diaphragm as a filtering medium, so as to form the chord of an arc of the interior of the barrel. The diaphragm or filter is made up of plates corrugated similarly to the ordinary filter-press plate, and perforated with holes every 4 or 6 inches square. These plates are supported on bearings bolted to the shell. On the top of the corrugated plates is placed the filtering medium—an open-woven asbestos cloth. It is about as coarse as the common gunny-sack, but the warp and woof are of much heavier thread.

Over this is placed an open grating, and the whole is held in place by cross-pieces, the ends of which rest under straps bolted to the inside shell. In this way, though the whole is rigidly held in place, it is very easily and quickly removed when necessary to change the asbestos cloth. Two valves on each end of the barrel above and below the filter, are provided for the inlet and outlet of the wash-water and solution respectively.

The barrel is charged by first filling the space under the filter with water, which at the same time is allowed to pass through the filtering medium and wash it, then the required quantity of water is put in above the filter. There are two methods of charging the pulp, lime-chloride, and sulphuric acid. In one the lime is so placed in the ore-charge in the hopper over the barrel that it goes in with the ore and is completely buried with it. The acid can then be added with very little danger of generating any gas before the plate on the charging-hole can

be put on and securely fastened. The other way, which seems still better, is to pour the acid first into the water, through which it sinks to the bottom in a mass, and does not mix. The ore is then let in, and the lime added last. The chances of generating any gas are stated in this way to be much less. A barrel so charged has been known to remain open 5 to 10 minutes after charging without generating gas, but it has been demonstrated that on the first revolution of the barrel the gas is at once liberated and creates considerable pressure. After the chlorination is complete the barrel is stopped, so that the filter assumes an horizontal position. The hose is attached to one of the outlet pipes, and conducts the solution to the reservoir tank. A hose is also attached to the inlet pipe, water is pumped in under pressure, and the leaching commences.

The air in the top part of the barrel is compressed, and forms an elastic cushion, which gives the wash-water perfect freedom to circulate over the whole surface of the charge, and wash every portion thoroughly with the smallest quantity of water possible. By washing in this way no gas is allowed to escape into the building. The solution runs into a covered reservoir-tank from which an exhaust fan draws the excess of gas, and discharges it outside the building. The length of time needed to do the leaching, varies with the leaching-quality of the ore. Charges have been leached in 40 minutes with a pressure of 30 to 40 lbs. per square inch. With higher pressures the time can be materially shortened. As can be readily seen, the ore in the barrel is in the best possible shape for rapid and perfect leaching. When the barrel is stopped, the ore settles on the filter, the coarsest and heaviest on the bottom, graduated evenly over the whole surface up through the charge to the slimes on top. In order to facilitate the leaching of charges carrying an excess of dust, a valve placed in the casing of the head (on a level with the surface of the pulp) is opened just after the barrel is stopped, and the slime remaining in suspension is run off into an outside washing filter-press, where it can be treated separately, and the charge washed in the usual way. The tailings are discharged into a car which will hold the whole charge of ore and water and then run out, or if water is abundant they are discharged into a sluice and washed away.

For leaching purposes the amount of water needed to wash a charge varies very little with the richness of the ore, going to show the perfect leaching-condition of the ore in the barrel. The amount required is about 120 gallons per ton more than the quantity used in the barrel for chlorination, which is about 100 gallons per ton.

In order to get a concentrated solution for after-treatment, and to

reduce the amount of solution to be treated, a tank is placed over the barrel, and when the richest of the solution and wash-water has run out into the reservoir-tank the discharge hose is connected with a pipe leading to the upper tank, and the washing is finished into it. The solution collected in this way is used in the next following charge in the barrel. The quantity of solution to be precipitated is thus reduced to 120 gallons per ton of ore treated.

The advantages claimed for this method are: (1) the freedom of the building from chlorine gas; (2) the control obtained over the perfect washing of the ore; (3) the small amount of labour, especially skilled labour, necessary; and (4) the small amount and simplicity of the machinery for the great amount of work accomplished.

One man of ordinary intelligence and a helper, are able to look after three barrels—charging, leaching, and discharging them. If the tailings are sluiced out no extra help is needed, but where they have to be trammed, one man in addition is necessary. The disadvantages are due to the necessary construction of the barrel, but do not interfere with its successful working. They are principally the amount of space taken up by the filter and the portion of the barrel underneath, and the fact that when the barrel is charged and running it is not perfectly balanced.

These disadvantages can be partly overcome by placing the filter close to the shell, only leaving sufficient space underneath to allow of free circulation, but bringing it up to the same height on the sides of the barrel as the horizontal filter; then, by using compressed air to displace the solution and wash-water, an equally good result can be obtained.

For the collection of the solution two tanks are necessary, each of ample capacity to hold a day's solution from all the barrels. Those for collection are placed on the same floor as the chlorinators (unless fall can be secured to place them below). On a step below are the precipitation-tanks, which should be of the same capacity and number as the collecting-tanks. The limit to size would probably be 50 tons capacity; where more is treated, another battery of tanks would be needed.

For a precipitant Mr. Rothwell recommends hydrogen sulphide gas generated from paraffin and sulphur or from iron sulphide and sulphuric acid as the cheapest and most satisfactory. It is generated, and then forced through the solution with a small air-pump, which at the same time forces air through, keeping the solution-tank in an agitated state and expelling any free chlorine gas. To save time the gas is turned into the tank while it is filling up, so that, when the tank is full, a few minutes finish the precipitation and collection. The tank is now allowed to stand two or

three hours, when it will have settled sufficiently to allow of the supernatant liquor being drawn off through a filter-press. There is little danger of precipitating arsenic and antimony which may be present when the process is worked cold, as they do not commence to come down till some time after the gold has been precipitated and collected.

This precipitant would not be, however, desirable with any considerable quantity of copper or lead in the solution, but small quantities can be dealt with in the after-treatment.

The loss in gold is considerably less if the precipitate is allowed to accumulate in the tanks, and a clean-up made after six or ten precipitations, than if it were filtered through a press and collected after every precipitation. There does not seem to be any advantage derived from a continuous precipitation and collection on this account if hydrogen sulphide gas is used, as the filters will soon become so coated and clogged with the sulphides as to retard rapid filtration without extreme pressure, which is sure to increase the loss.

The handling of a large number of filter-cloths is also a source of loss, no matter how carefully done. The asbestos filter-cloth under ordinary conditions will last 100 charges, and can be changed in about an hour and a half; one cloth has been known to last 150 charges.

The life of the supporting plates and grating can be made to equal the life of the lining of the barrel, and with barrels that have several thousand charges to their credit this shows little wear.

The latest suggested modification of the chlorination process appears to be that patented by W. D. Bohm, described and illustrated in the *New York Mining Journal* of December 31st, 1892. Briefly stated, the principles involved in it, are a forced upward circulation of the solvent solutions through the powdered ore placed in a suitable vat, of special construction.

The circulation is maintained until the precious metals are dissolved, when air-pressure is applied above the charge, to force out as much of the solution as possible, wash-water being subsequently forced up from below, or admitted above the charge, and then forced out in a similar manner, the previous constant flow upwards having caused (it is stated) such a deposition of the sand as to allow the liquid to be expressed rapidly and cleanly. It is claimed by using a solution of chlorine in water, and circulating it in closed pipes and vessels, that a considerable saving of gas is effected, and that as rapid results are obtained as with agitation, with much less power and wear and tear, while ores of different character can be treated in a short time and with little labour.

The plant is said to be equally applicable to cyanide treatment; the

precipitation of the gold being effected on shavings of an alloy of zinc and sodium is claimed to be more efficient than zinc alone. The chlorine water is circulated through the ore in a manner similar to a cyanide solution. It is claimed for the electrolytic chlorine-producing apparatus, the invention of two Russian chemists (which is worked in connexion with this process), that a great reduction in the cost of producing chlorine is effected; that common salt will supply the place of sulphuric acid and chloride of lime, that the machine is simple, requiring no skilled labour, and has been running with success at the El Dorado mine, Siberia.

This plant, in charge of an engine-driver, produces 40 lbs. of chlorine gas from less than 150 lbs. of common salt, utilizing 5 horse-power. The results are vouched for by M. Leon Perret, mining engineer to the Imperial Russian Government.

A diaphragm-pump of special construction prevents any emulsified grease or other undesirable matter from getting into the solution, and stops leakage of the gold solution through defective glands, etc.

The cost of working this process is stated to vary with the locality. Kaolin ores of Mount Morgan have been treated before the erection of the electrolytic chlorine generator at the following cost :—

	s.	d.
10 lbs. of bleach, at 4 cents	1	8
8 „ acid, at 2 cents	0	8
Labour and power	1	5½
	3	9½

The time occupied for each charge being 4 hours.

The extraction is not stated, but a proportionate increase in cost of chemicals would be necessitated in the treatment of ores requiring more chlorine.

Mr. Claude Vautin mentions, as recent useful innovations, glass-lined iron pipes, manufactured by Messrs. D. Rylands & Co.; enamelled iron diaphragm-pumps for conveying corrosive solutions, made by Messrs. Scott & Sons; and a compound containing upwards of 75 per cent. of available sulphuric acid, solid below the temperature of boiling water, which can be packed in iron drums, and shipped as ordinary cargo (obtainable from the Ore Dressing Co., 42, Old Broad Street); also chloride of lime packed in specially prepared iron drums. The form in which these two latter reagents are sold may be of utility to persons employing the chlorination process, especially in localities distant from a railway where transport has to be taken into account.

Mr. T. R. Rose,* commenting on Mr. Vautin's statement "that precipitated sulphide of copper is not to be recommended as a practical precipitant for gold, owing to its physical condition, and the facility with which it is oxidized to sulphate of copper," says:—"In accepting this statement as correct in a certain limited sense, it must not be forgotten that the suitability of the precipitant depends mainly on the manner in which it has been prepared. By adding a boiling saturated solution of sulphide (or rather a mixture of the polysulphides of sodium) to the equivalent proportion of a boiling saturated solution of sulphate of copper with vigorous stirring, sulphide of copper is precipitated in a granular form, which settles quickly in water. This substance is more active in the precipitation of gold than the fused sulphide. Its activity is due to the larger surface presented by the porous granules, and to the fact that it is easily decomposed. It is, as Mr. Vautin says, somewhat liable to atmospheric oxidation, but the loss, owing to this cause, is insignificant, because it is always kept covered with water, and can only take up such oxygen as may be dissolved in the latter. The granules of the sulphide of copper can be made the size of peas, under favourable conditions, and there is no practical difficulty met with in precipitating gold rapidly and completely by their use. A point in favour of the fused sulphide of copper is the readiness with which it can be obtained, and as it has been proved to answer well, there is no reason why it should be rejected in favour of the precipitated salt."

The Cyanide Process.

The plant required for the MacArthur-Forrest process consists of crushers (stamps or rolls) to pulverize the ore to 20 to 60 mesh, and leaching and precipitating-tanks of much the same description as those used for chlorination: but as its professed object is the treatment of the ore in bulk, the number of tanks, etc., required for this method of treatment are of necessity much larger than in chlorination works. The cost of erection of such a plant on a basis of treating 50 tons per day would not, so it is said, ordinarily exceed £6,250, everything included.

The report of Mr. J. R. Bradshaw (assayer, etc., of Charters Towers) states, without going into details, that from information he received at Ravenswood, North Queensland, it can be worked there to profit at a cost of £2 per ton, but the cost must of necessity depend on the district and surrounding circumstances.

Mr. Bradshaw claims, moreover, economy for the process in regard

* *Mining Journal*, March 11th, 1893, page 276.

to time occupied in treatment, as compared with raw amalgamation in pans, stating that with an ordinary wheeler, ½ ton of auriferous stone of about 2½ ounce grade is decomposed in 6 hours, representing the capacity of one wheeler as being equal to treating 12 tons per week. Mr. McIntyre, the company's manager at Ravenswood, he goes on to state, is erecting two wheeler pans, measuring 5 feet in diameter, capable of carrying a charge of 3 tons of ore each, and it will thus be seen that with two pans, 24 tons of ore can be decomposed in 24 hours.

It would be interesting to learn how those arithmetical calculations panned out in actual practice as regards power consumed and other details.

A test of the Ravenswood ore made by Mr. Bradshaw gave an extraction of 89 per cent. of the gold and 70 per cent. of the silver.

If the success of the process demanded the mechanical incorporation of the cyanide solution with the ore in pans, it would seem likely to be foredoomed to failure, and on this account the idea is now almost entirely abandoned.

It would probably be found that, for convenience and speed, barrels could be advantageously substituted for pans, but modern developments of the process have done away with the original notion of auxiliary agitation.

The Chemistry of the Cyanide Process.

This has been ably dealt with by Messrs. Charles Butters, Ph.D., and J. E. Clennel, B.Sc., in a series of papers which appeared in the *New York Mining Journal*, in October last, from which the following extracts are taken:—

Elsner has shown that the very finely divided gold obtained by precipitating a solution of the chloride with ferrous sulphate may be dissolved by potassium cyanide, provided there is an excess of oxygen present.

The compound formed may be looked upon as a double cyanide of gold and potassium (KCyAuCy), and the reaction which takes place may therefore probably be represented by the following equation:—

$$2Au + 4KCy + O + H_2O = 2KAuCy_2 + 2KHO.$$

Two interesting points are indicated by the above equation, which it is well to bear in mind, in applying potassium cyanide as a gold solvent on a commercial scale:—

1. That the quantity of cyanide theoretically capable of dissolving a given amount of gold is infinitesimal, compared with the weight actually required in practice. Taking Au = 196·8,

K = 39·04, and Cy = 25·98, it will be observed that 130·04 parts by weight of potassium cyanide, should be capable of dissolving 196·8 parts of gold, or approximately two parts of the cyanide salt should dissolve three parts of gold. The minimum actual consumption in treating free-milling ore, assaying, let us say, 10 dwts. per ton, is about 3 lbs. per ounce of gold recovered, or roughly forty parts by weight of cyanide for one part of gold. In the leaching-tanks alone 1 lb. of cyanide is generally consumed per ton of material treated.

2. That an extremely small quantity of surplus oxygen is sufficient to cause the solution of the gold, 15·96 parts being required for 393·6 parts of gold, or say as 1 : 25. The air present in a porous mass of tailings, with that dissolved in the water used in making up the solution, is in fact more than ample to effect the reaction. To explain the enormous excessive consumption of cyanide, we must bear in mind the great instability of the simple cyanides.

Hydrocyanic acid is liberated from its salts by all mineral acids, carbonic acid, and all common organic acids. Atmospheric carbonic acid is sufficient to set up a certain amount of decomposition, in which a constant evolution of hydrocyanic acid takes place as follows:—

$$2KCy + CO_2 + H_2O = 2HCy + K_2CO_3.$$

Then, again, we must consider the proneness to oxidation which the cyanates exhibit, and which in fact lies at the base of most of their technical applications. Potassium cyanide readily changes into cyanate and ultimately into carbonate:—

$$KCN + O = KCNO;$$

$$2KCNO + 3O = K_2CO_3 + CO_2 + N_2.$$

The presence of alkalies, which always occur in commercial cyanide, tends to induce the peculiar and little understood decomposition termed hydrolysis, which seems to be mostly produced in the zinc boxes by the presence of that metal.

In the reaction alluded to above, the alkali appears to determine a chemical change in which water plays a part, while the alkali itself is not in the least affected.

There are good grounds for supposing that in dilute solutions a dissociation of the cyanide takes place, so that what we term a weak solution of potassium cyanide is in reality a mixed solution of potassium hydrate and hydrocyanic acid:—

$$H_2O + KCy = HCy + KHO.$$

This being the case, it is only natural that hydrocyanic acid should be constantly given off from all vessels in which weak cyanide solutions are freely exposed to the air, and its smell is, in fact, generally noticeable in the neighbourhood of the tanks in which it is stored.

The consumption of the reagent is on these grounds evidently enhanced by the agitation or circulation systems, since these methods involve a constant exposure of fresh surfaces.

Another source of waste is due to the tendency of the simple cyanides to form double salts with each other, or with other metallic compounds.

Salts of iron, and to a lesser extent of aluminium, magnesium, calcium, and the alkali metals are liable to occur in the tailings, especially after long exposure to atmospheric influences.

It seems, therefore, that under the most favourable circumstances an enormous waste of cyanide must take place, which may partly, however, be mitigated by the use of closed tanks and careful attention to the purity both of the cyanide itself and the water employed to dissolve it.

Action of Cyanide on Pyritic Material.—Various additional decompositions take place, when cyanide is applied to the treatment of pyritic ores or tailings. The surface ores of the celebrated banket formation of South Africa consist almost exclusively of silica and oxide of iron, the silica occurring in the form of rounded pebbles, embedded in a softer matrix highly charged with ferric oxide, which gives it a characteristic reddish tinge. The gold* is found in this matrix associated with the oxide of iron, or sometimes in small scales on the surface of the pebbles. The pebbles themselves carry little or none.

At a lower level this free-milling banket passes into an ore precisely similar in structure, but much harder, and containing the iron in the form of sulphide instead of oxide, which gives it a peculiar bluish tint.

There can be no doubt that the free-milling ores have been formed by the gradual oxidation of the pyrites through the influence of air and moisture during a long period of time, and in fact we see this change in progress, wherever pyritic material has been exposed to the action of the atmosphere. The first effect observed is the conversion of ferric sulphide

* Taking into account the production from tailings and concentrates the average yield of the Witwatersrandt ore was 12 dwts. 5 grains per ton in 1891 and 12 dwts. 13 grains in 1892. At the end of 1891 there were 1,540 stamps in operation, which yielded 11·23 dwts. per ton by direct amalgamation, whilst the average of free-gold saved fell in 1892 to 9·87 dwts., 2,035 stamps being in operation at the close of the year; the total yield having increased rather than fallen off goes to show that though the ore with depth has become more sulphuretted, it has so far been successfully treated.

into a soluble sulphate, free sulphuric acid being liberated. By the action of the air again on the ferrous sulphate, certain insoluble basic sulphates of variable and somewhat complex composition are found to result, whilst a certain amount of soluble ferric sulphate is likely to be produced at the same time.

The pyritic ores of the Witwatersrandt contain also small amounts of copper, arsenic, and sometimes cobalt and nickel, but the amount of these foreign metals has hitherto been so small that it has not practically interfered with the process.

As the fact has been observed, however, at the Robinson chlorination works, that copper and arsenic seem to occur in gradually increasing quantities with the increasing depths of the mines from which these concentrates were purchased, it is possible that these elements may be a serious source of trouble in the future.

If one attempts to treat a charge of partially oxidized pyritic tailings directly with cyanide solution, the free sulphuric acid present which renders the moisture they contain distinctly acid, sets free hydrocyanic acid. Ferrous sulphate (green vitriol) reacts upon the cyanide with the formation of ferrous cyanide, a yellowish-red flocculent precipitate :—

$$FeSO_4 + 2KCy = FeCy_2 + K_2SO_4.$$

This, however, under ordinary circumstances is slowly converted into potassium ferrocyanide by the excess of cyanide present :—

$$FeCy_2 + 4KCy = K_4FeCy_6.$$

If sufficient acid be present, the ferrocyanide reacts on an additional quantity of the ferrous salt, ultimately giving rise to a blue precipitate or coloration (Prussian blue) :—

$$3K_4FeCy_6 + 6FeSO_4 + 3O = Fe_2O_3 + 6K_2SO_4 + Fe_7Cy_{18}.$$

A coloration of that sort on the surface of the tailings or in the solution is therefore a sure indication that acid iron salts are present, and that a large waste of cyanide is taking place.

Ferric salts, when present unmixed with any ferrous compounds, decompose the cyanide solution with evolution of hydrocyanic acid, and precipitation of ferric hydrate :—

$$Fe_2(SO_4)_3 + 6KCy + 6H_2O = Fe_2(OH)_6 + 6HCy + 3K_2SO_4.$$

This reaction takes place in two stages, the first being the formation of a soluble but very unstable ferric cyanide, giving a dark brown solution :—

$$Fe_2(SO_4)_3 + 6KCy = Fe_2Cy_6 + 3K_2SO_4,$$

which decomposes as follows :—

$$Fe_2Cy_6 + 6H_2O = Fe_2(OH)_6 + 6HCy.$$

This gives rise to ferric hydrate, part of which, being in a finely divided

colloidal condition, is with difficulty removed, as it chokes the pores of the filters.

A mixture of ferrous and ferric sulphates, such as is probably always present in partially oxidized pyritic tailings, causes the appearance of a blue colour on the addition of cyanide, after the free alkali of the commercial product has been neutralized, Prussian blue (ferric-ferrocyanide), $Fe_4(FeCy_6)_3$, being produced when the ferric salt is in excess, and Turnbull's blue (ferrous-ferricyanide), $Fe_3(FeCy_6)_2$, when the ferrous salt predominates.

Before attempting to treat pyritic material or products with cyanide, it is necessary therefore to get rid of the free sulphuric acid and soluble iron compounds. This is generally done by leaching with water until the liquid running off the tanks no longer shows a coloration with ammonium sulphide. After this treatment the insoluble basic sulphates which still remain, and being gradually decomposed by water, would act upon the cyanide solution, are dealt with by washing with caustic soda or lime water. This converts the basic salts into ferric hydrate and sodium or calcium sulphates. When the quantity of free acid and iron salts is small, the preliminary wash-water may be advantageously omitted.

Lime in the dry state is sometimes mixed with the tailings before the cyanide treatment commences. When this method is adopted the iron is precipitated as a mixture of ferrous and ferric hydrates. After the washing with alkali is completed, the tanks are allowed to drain, and strong cyanide solution of about 6 per cent. is pumped on.

Even after this treatment the consumption of cyanide with moderately pyritic tailings, which have been partially decomposed by exposure, is found to be four times that which occurs with free-milling material.

The presence of a large excess of alkali in the solution brings about various secondary reactions, which lead to a loss of cyanide, such as the hydrolysis before referred to, and a peculiar action in the zinc box mentioned later.

Lime though slower in its action is preferable to caustic soda as a neutralizing agent, as it is equally effective in decomposing the iron salts, less active in producing secondary reactions on the cyanide, and also less energetic in attacking the zinc in the precipitating-boxes.

Ferric hydrate does not appear to be acted upon by potassium cyanide, but ferrous hydrate, which is formed on the neutralization of the iron salts by alkalies, reacts on the cyanide in excess, with the formation of ferrocyanide of potassium :—

$$Fe(OH)_2 + 6KCy = K_4FeCy_6 + 2KOH.$$

Deposition of Gold from Cyanide Solutions.—Under certain conditions, such as the absence of sufficient oxygen in the solution, a partial precipitation of the previously dissolved gold appears to occur. If by any chance the solution should become acid, there is a decomposition of the double cyanide of gold and potassium, in which the gold is generally supposed to be thrown down as (insoluble) aurous cyanide :—

$$KAuCy_2 + HCl = KCl + HCy + AuCy.$$

In working on the circulation-and-transfer system it is found that where pyritic material is under treatment it is not safe to transfer a solution already rich in gold to a fresh lot of tailings, as the extensive decomposition of the solution which takes place may lead to a final loss of gold.

Selective Action of Cyanide.—It is claimed by the inventors of the MacArthur-Forrest process, that in a mixture containing metallic gold, silver, copper, and base metals, cyanide of potassium exerts a selective action, dissolving first the gold, then the silver, and afterwards attacking the copper and baser metals.

The process, however, does not appear to have been successfully applied to ores, such as are met with in parts of California and Australia, containing considerable quantities of foreign metals.

Ores containing sulphides of silver and copper produce considerable decomposition of cyanide, the copper being partially dissolved as sub-sulpho-cyanide, the silver, however, remaining unattacked.

In two experiments carried out by Mr. Wm. Bettel, chief chemist of the Robinson Gold Mining Company, on an ore from the Albert silver mine (containing 30 ounces of silver per ton and 10 per cent. of copper), it was found that no extraction of silver occurred, this metal being present as sulphide.

Action of the Zinc Shavings on the Solution.—We must now consider the action of the zinc on the gold cyanide solution. Theoretically, a simple substitution of zinc for gold occurs in accordance with the following equation :—

$$2KAuCy_2 + Zn = K_2ZnCy_4 + 2Au.$$

Taking Zn = 65·1, Au = 196·8, it follows that 65·1 parts by weight of zinc should be sufficient to precipitate 393·6 parts of gold, or 1 lb. of zinc should precipitate 6 lbs. of gold. The actual consumption of zinc is about 1 lb. per troy ounce of gold recovered. It is evident, then, that zinc is consumed in some other way than in mere replacement of gold.

During the passage of the solution through the zinc boxes a constant vigorous evolution of small bubbles may be noticed, which are found to consist chiefly of hydrogen. The outflowing liquid is found to possess a

greater degree of alkalinity than it had on entering at the head of the box, and a smell of hydrocyanic acid and sometimes of ammonia is constantly observed in the neighbourhood of the boxes.

It is clear, then, that a decomposition of the potassium cyanide solution itself by the zinc is in progress, and this is not to be wondered at when we consider the powerful electro-chemical effect which must be produced by the contact of such a highly positive metal as zinc with a strongly negative metal such as gold.

Ordinary commercial zinc loses weight when immersed for some time in cyanide solution, but the action is slow. It is doubtful whether pure potassium cyanide would have any action at all on chemically pure zinc. It is well-known that the copper-zinc couple produced by immersing zinc in a solution of a copper salt decomposes water.

An analogous reaction of the zinc couple accounts for the evolution of hydrogen above-mentioned :—

$$Zn + 2H_2O = 2H + Zn(OH)_2.$$

The hydrate of zinc is at once dissolved in the excess of cyanide :—

$$Zn(OH)_2 + 4KCy = K_2ZnCy_4 + 2KOH,$$

which reaction accounts for the increased alkalinity of the solution.

There are reasons for believing that the black deposit formed on the zinc shavings is an actual chemical compound of gold and zinc, which acts as the negative element in the electric couple, the undecomposed zinc forming the positive element.

When strong solutions of caustic soda have been used for neutralizing the acid salts of the ore, a white deposit is frequently observed on the zinc. The alkali first attacks the metal to form a zinc-sodium oxide :—

$$Zn + 2NaHO = Zn(ONa)_2 + 2H.$$

This then reacts on the double cyanide of zinc and potassium always present in the solution, and precipitates the white insoluble simple cyanide of zinc :—

$$2H_2O + Zn(ONa)_2 + K_2ZnCy_4 = 2ZnCy_2 + 2NaOH + 2KOH.$$

This reaction is of some importance as affording one means by which the excessive accumulation of zinc in the solutions is avoided.

Affinity of Zinc for Cyanogen.—Potassium auro-cyanide ($KAuCy_2$) appears to be one of the most stable of the salts of gold, and the reaction in the zinc boxes, shows that the affinity of zinc, together with potassium, for cyanogen, is greater than that of gold with potassium for the same radicle. Hence a solution of potassium cyanide cannot dissolve gold which is in contact with zinc, neither can gold replace zinc in a solution of the double cyanide of zinc and potassium. So long as any zinc is present,

therefore, we need not fear that the precipitated gold will redissolve in the excess of potassium cyanide flowing through the boxes.

It is evident also that the cyanogen contained in the double cyanide of zinc and potassium is not available for dissolving gold, nor when a solution charged with zinc is employed in the treatment of a fresh lot of tailings it is only effective in so far as it contains a certain quantity of simple cyanide of potassium or other alkaline cyanide.

New Methods of Precipitation.—The cyanides of sodium and ammonium and those of the alkaline earths (calcium, barium, etc.) will dissolve gold as well as potassium cyanide. Sodium cyanide is more difficult to manufacture than the potassium compound, but a given weight of it should be more effective than the same weight of potassium cyanide, inasmuch as 49 parts of the former are equivalent to 65 of the latter.

The advantages of Molloy's process and others which employ sodium or potassium amalgam will be referred to later. The alkali metal is obtained by the electrolysis of the carbonate between electrodes of lead and mercury :—

$$Na_2CO_3 = Na_2 + CO_2 + O.$$

The sodium forms an amalgam with the mercury. Sodium amalgam may also be manufactured direct from its elements. It is claimed for this method of precipitation that the whole of the cyanogen is restored to a condition available for dissolving gold as shown by the reaction :—

$$Na + KAuCy_2 = Au + KCy + NaCy.$$

Composition of the Zinc Slimes.—Any base metals which happen to be in solution in the cyanide liquor are liable to be precipitated by the zinc along with the gold. Hence the zinc slimes are found to contain a certain percentage of copper as well as traces of arsenic and antimony. Moreover, any impurities in the zinc will also find their way into the slimes, as zinc will be dissolved by the cyanide in preference to any less oxidizable metals, *e.g.*, tin and lead. Silver is dissolved by the cyanide and re-precipitated by zinc by a set of reactions precisely analogous to those of gold :—

$$2Ag + 4KCy + O + H_2O = 2KAgCy_2 + 2KOH, \text{ and}$$

$$2KAgCy_2 + Zn = K_2ZnCy_4 + 2Ag.$$

It has been observed that the proportion of silver to gold is greater in the cyanide bullion than in the gold from the batteries, and this is explained by supposing that the loss of silver in amalgamation is greater than that of gold.

Treatment of the Zinc Slimes.—The removal of the zinc is a troublesome operation, and is only very partially carried out in smelting the

dried slimes. The admixture of sand is made for the purpose of forming a fusible silicate of zinc.

A portion of the zinc is volatilized and burns at the mouth of the crucible with a greenish flame, producing the white oxide (ZnO), which is found encrusting the flues, and no doubt carries with it no inconsiderable quantity of gold and silver. The most promising method of treating these slimes appears to be that suggested by Mr. Bettel of fluxing with acid sulphate of soda and fluorspar.

Attempts to remove the zinc prior to smelting have been only partially successful, as all such methods involve the filtration of a slimy mass, which retains soluble salts with great tenacity.

The slags from the fusion of the zinc slimes contain a considerable amount of gold, some of which is in the form of round shot, and may be removed by pounding up the slag, passing through a coarse sieve, and panning off. The residue from the first fusion should always be fused again with additional lead to form an alloy with the gold. The same lead bars may be used for a number of successive fusions of the slag, and when sufficiently enriched the gold can be recovered by cupellation.

Testing of Cyanide Solutions.—It is a matter of importance to determine exactly what strength of cyanide solution is used in treatment of tailings. The ordinary method of testing depends on the fact that silver cyanide is soluble in excess of potassium cyanide, with the formation of a double cyanide of silver and potassium :—

$$KCy + AgNO_3 = AgCy + KNO_3, \text{ and}$$
$$AgCy + KCy = KAgCy_2.$$

When silver nitrate solution is added drop by drop from a burette to a solution of cyanide a white precipitate is formed, which quickly redissolves. At a certain stage the precipitate becomes permanent, when, in fact, the whole of the cyanide has been converted into the soluble silver salt, and an additional drop of silver nitrate produces a permanent precipitate of the insoluble simple cyanide of silver :—

$$KAgCy_2 + AgNO_3 = KNO_3 + 2AgCy.$$

From these reactions 107·66 parts by weight of silver are equivalent to 130·04 parts of potassium cyanide. A convenient standard silver solution is one of such strength that every cubic centimetre added to 10 cubic centimetres of the solution to be tested, corresponds to 0·1 per cent. of pure potassium cyanide.

This method gives good results when pure cyanide solutions are under examination, but when the solutions contain zinc it is difficult if not impossible to determine the end of the reaction. A white flocculent pre-

cipitate occurs at a certain stage, probably consisting of simple (insoluble) cyanide of zinc, formed by the decomposition of the soluble double cyanide.

This precipitation occurs long before the whole amount of potassium cyanide has been converted into the soluble double salt of silver ($KAgCy_2$) for the solution after the appearance of the flocculent precipitate still gives the Prussian blue reaction with acidulated ferrous sulphate.

A standard solution of iodine in potassium iodide may be used with great accuracy for determining the total amount of cyanogen in a solution whether in combination with zinc or not, making use of the reaction :—

$$KCy + I_2 = KI + ICy.$$

The colour of the iodine is discharged so long as an excess of cyanide is present. The sharpness of the terminal reaction may be increased by adding a small quantity of starch to the solution under examination, which gives a permanent blue colour as soon as an excess of iodine has been added.

What is most needed, however, is a rapid method of determining the amount of cyanide available for dissolving gold, for, as was pointed out above, the cyanide in combination with zinc is not available for that purpose.

The method of testing solutions containing zinc for available cyanide which was introduced by Mr. Bettel at the Robinson Company's works is as follows :—Two perfectly clean flasks of equal size are taken. To each of these is added a considerable bulk, say 50 cubic centimetres of the solution to be tested and 50 cubic centimetres of water. The liquid in both flasks will probably appear slightly turbid, but the degree of turbidity will be the same in each. Standard silver nitrate solution is run into one flask until the slightest possible increase in turbidity is observed in comparison with the liquid in the other flask. This point is taken as indicating the conversion of the whole of the free potassium cyanide into the soluble silver salt, and therefore as determining the amount of available cyanide present.

The amount of gold in the solution is generally found by evaporating a known bulk with litharge, fluxing the residue and cupelling the resulting lead bullion. Evaporation on lead foil may likewise be employed.

Poisonous Properties of Cyanide.—A few words may not be out of place in this connexion. Although one of the most rapidly and deadly of known poisons when taken internally, its action as a blood poison is much less violent. Nevertheless, when introduced into cuts it produces very painful sores. The men employed in cleaning-up and in melting the slimes are subject to a peculiar eruption, especially on the arms, and complain of headache, giddiness, and general depression.

Ferrocyanide of potassium has been recommended as a remedy for the eruption; it may be taken internally and also applied as a lotion. Considering the dangerous nature of the reagent it is remarkable how few fatal cases have occurred through the use of cyanide on a large scale. In cases of poisoning precipitated carbonate of iron, obtained by mixing solutions of sodium carbonate and ferrous sulphate, may be used as an antidote, forming internally an insoluble blue compound with the cyanide.

Hydrocyanic acid acts directly on the nervous system, causing instant paralysis, hence any treatment which will excite the action of the nerves such as applications of cold water to the spine, inhalation of ammonia, etc., may be tried in cases of faintness produced by breathing the acid vapour.

The disposal of waste cyanide liquors is a matter of serious consideration. Solutions containing 0·1 to 0·2 per cent. of potassium cyanide must occasionally be discharged and are likely to contaminate the water of the dams or streams which receive them to a dangerous extent. If some effective means of precipitating the zinc, or better still of dispensing with the use of zinc altogether, could be devised there would never be any necessity for allowing cyanide liquors to leave the works.

Applications of the Cyanide Process.

Mr. Almarin B. Paul appears to have originated the idea of using cyanide solution in the battery crushing the ore wet. This he claims to have done with success at the Calamut mill, Shasta County, California. He states that his plan is in all cases to crush with a weak solution, and should the ore require a higher percentage of cyanide when the first solution has percolated below the surface of the ore, after the tanks have been filled, a stronger one can, if necessary, be introduced. All the dust and disagreeable effects of dry-crushing is thus avoided. The loss of cyanide in crushing is but nominal, and is off-set by the cheapness of working and completeness of distribution of the cyanide through the pulp in the tanks. This plan has been followed of late at the May Consolidated works in South Africa, but it does not appear to be applicable to all cases, as generally when ores have been crushed wet and run direct to the tanks the pulp has packed so hard as to be impenetrable to the leach liquor. Ores containing impurities, such as arsenic and tellurium, have proved obstinate to deal with.

The consumption of available potassium cyanide may for practical purposes be divided into avoidable and unavoidable decomposition; each of these factors varying with the composition of the ore, the strength of the solution, time of contact, and method of treatment.

Tests made on a heavy, raw sulphide gold ore by Mr. C. W. Merrill, using a 2 per cent. solution, showed that it was capable of decomposing available cyanide at the rate of 12 lbs. per ton in 24 hours, but that by taking all possible precautions this loss could be reduced to 3 lbs. per 24 hours.

The reduction in loss of cyanide seems, however, to have been set off by an extravagant loss of time, as it is necessary to continue the treatment for 7 to 10 days.

A series of extraction tests proved that a $\frac{1}{4}$ per cent. solution when reinforced each day gave as good results as a stronger one, and with a decomposition of only 5 to 7 lbs. of cyanide per ton. As the ore contained acid salts of iron and fine copper sulphides, both of which decompose potassium cyanide, the probability is that the weak solutions were rendered inert after a few hours' contact with the ore, hence the necessity of reinforcing them.

A series of experiments, published in detail in the *New York Mining Journal* of December 24th, 1892, were made by Mr. G. E. Kedzie, M.E., to determine how far the gold-bearing pyritic ores of Ouray, Colorado, could be successfully treated by the cyanide process, and also to learn the conditions under which the most complete extraction could be obtained, together with the amount of cyanide consumed. These experiments throw much light on the question of how far it is practicable to treat ores of a similar description by the cyanide method.

The samples under treatment contained gold and silver in varying proportions, ranging from 0·27 ounce to 145·90 ounces of the former, and 0·65 ounce to 458 ounces of the latter metal, the value of the high-grade ores and tailings being determined by triplicate scorification assays, and that of the low-grade ores and tailings by duplicate crucible fusions. The gangue in some cases was quartzose, in others calcareous or clayey, and contained iron pyrites, copper pyrites, and magnetite, either alone or in admixture.

The deductions drawn from these tests were: (1) the gold is more readily extracted than the silver; (2) under the same conditions the percentage of extraction is increased—(*a*) by the fineness of the pulp—(*b*) by the duration of treatment—(*c*) by the strength of the cyanide solution; (3) the greater the amount of cyanide added to the ore, the higher will be the percentage of extraction, but in this case the total values extracted for each pound of cyanide consumed are less than when a smaller amount of cyanide is used for each ton of ore treated; and (4) when the same amount of cyanide is used for each ton of ore treated the percentage of extraction is greater when the weight of the solution is equal to that of the ore taken.

His conclusions are that there are no flattering indications of the process being a metallurgical success with the pyritic ores under consideration. With high-grade ores, which are under no circumstances adapted to this process, the percentage of extraction under the most favourable circumstances is low, lower even than the results obtained by amalgamation. However, the total values extracted for each pound of cyanide consumed are relatively high.

With low-grade ores, even where the low value of the tailings will admit of their being thrown away, the total values extracted in a majority of instances are less than the cost of cyanide consumed, to say nothing of milling expenses. In these experiments the percentage of potassium cyanide varied from 0·5 to 1·5 per cent. The treatment lasted from 12 to 60 hours. The extraction of the gold ranged from 2·31 to 84·62 per cent. The extraction of the silver varies from *nil* to 88·88 per cent. The proportion of the potassium cyanide added per ton of ore represented from 5 to 60 lbs., and its consumption rose from 3·2 to 50·8 lbs. per ton. The lowest value extracted per lb. of cyanide consumed was 0·28 dollar and the highest 53·72 dollars. The latter result was obtained in treating an ore which assayed 145·90 ounces in gold and 458 ounces in silver per ton.

Mr. Kedzie suggests, as the outcome of his observations, that the strongest solutions used should not exceed in strength $\frac{1}{2}$ per cent. of potassium cyanide, and good results are anticipated with much weaker ones. Arrangements should be made for the slow percolation of the cyanide solution, so as to keep the ore always in contact with a solution of nearly normal strength, and finally the period of lixiviation should be extended to 96 hours, if necessary, in order to secure a more complete extraction. To comply with these conditions the mechanical difficulties of leaching and loss of time must evidently be debited against a higher return.

Messrs. Clennel and Butlers state* that the cyanide process is the only method which has hitherto met with success in treating the tailings derived from various South African mills after amalgamation. The adoption of the process has been rapid, and its success most remarkable. Despite its simplicity on paper, the working metallurgist who attempts to carry out the process on a large scale soon finds himself confronted with difficulties. These arise from the nature of the material under treatment, and from the manipulations necessary in applying the treatment to ore in bulk.

Neither the solution of the gold in the ore under treatment, nor its precipitation, nor the conversion of this precipitate into bullion, is perfect

* Continuation of articles in the *New York Mining Journal* before referred to.

theoretically or practically. Losses occur in each operation, and the consumption of cyanide and zinc has been shown to be much in excess of what is indicated by the various chemical reactions primarily involved.

Hitherto the process has only been successfully applied to those ores or tailings usually described as free-milling, *i.e.*, such as are capable of yielding the greater part of their gold in the ordinary amalgamation process. The promoters of the process have been singularly fortunate in South Africa in the nature of the material they have had to deal with. The ores forming the upper portion of the main reef series, extending to a depth of 10 to 150 or 200 feet (as on the Robinson property) consist almost exclusively of silica and oxide of iron. They contain practically no substance, except gold and silver, which the cyanide is capable of attacking.

It is in dealing with ores or products containing sulphide of iron, especially when these are partially oxidized to sulphates and those containing compounds of lead, zinc, etc., that the difficulties of the process begin.

Solution of the Gold.—When the cyanide process was first introduced, about two years ago, it was thought necessary to agitate the material under treatment with the cyanide solution to obtain a satisfactory extraction. It was soon found, however, that the power needed and the rapid decomposition of the reagent were items of expense, which more than compensated for the greater percentage of the precious metals obtained in comparison with simple lixiviation. The operation is carried out by the African Gold Recovery Company, who represent the patentees in South Africa, as follows :—The damp tailings from the settling-pits are charged into wooden vats holding 35 to 50 tons, originally built square. The best works are now building them circular, those at the Robinson works holding 75 tons, and at the Langlaagte Estate 400 tons, whilst at the New Primrose still larger ones are being constructed.

The vats are filled to within a few inches of the top, and the surface of the tailings levelled. Cyanide solution of 0·6 to 0·8 per cent. strength is then allowed to flow into the tank until it is entirely filled.

The ore settles from 3 inches to 1 foot below the rim of the tank (the amount of shrinkage depending on the depth of the vat). This solution is allowed to remain undisturbed in contact with the ore for 12 hours.

Each vat is provided with a false bottom (usually a wooden framework covered with cocoanut matting). Below this is a layer of coarse sand and pebbles through which the solution percolates.

An iron pipe communicates with the vat below the false bottom, and conveys the filtered solution to the zinc boxes, where the precipitation takes place. After 12 hours' contact with the ore, the solution is allowed to drain out of the tank by opening a cock.

The dilute cyanide solution does not attack the wood, or corrode the iron piping to any appreciable extent. In wear and tear of the apparatus the use of cyanide, therefore, offers advantages over chlorine, which otherwise might be applied just as cheaply to the treatment of these oxidized surface-ores. Brass plungers and pump-valves are attacked, but not very rapidly. The pumps at the Robinson works stood for four months with comparatively little wear, but iron is preferable for the pump fittings where cyanide is employed.

As the liquor is drawn off during the leaching process it is replaced by fresh solution. This operation is continued for a longer or shorter period, depending on the grade of the tailings, for 6 to 12 hours. At the end of this period, known as the strong solution-leaching, a weaker solution (containing 0·2 to 0·4 per cent. of cyanide) is turned on, and allowed to filter through the ore for about 8 to 10 hours. This weak solution is then drawn off through another zinc box (known as the weak zinc box). Finally, a quantity of water is run into the tank, more or less equivalent to the amount of moisture which the ore contained when the tank was filled. This last washing-water displaces the weak cyanide solution, so that the volume of cyanide solution in use remains unchanged. The weak solution is in fact the liquor which has previously passed through the zinc boxes into the receiving-tanks or sumps, and has been pumped again to the leaching-tanks. The cyanide is usually supplied in cases containing 170 to 195 lbs. of crude cyanide, contaminated with carbonaceous matter and iron, but containing 72 to 78 per cent. of pure potassium cyanide.

This cyanide is usually dissolved in a small volume of water to form a highly concentrated solution, a special tank being employed for this purpose. The solution of required strength is obtained by adding this concentrated solution to the dilute liquor in the sumps. By this means a dilute solution of a given strength is more accurately arrived at than if the cakes of cyanide were dissolved directly in the required volume of water, since the percentage of cyanide can be more accurately determined in a strong solution. The actual amount of solution used is about half a ton of the stronger (0·6 to 0·8 per cent.) and half a ton of the weaker (0·2 to 0·4 per cent.) solution for every ton of ore leached.

When the final wash-water has been added, and has displaced the weak solution, the residues are discharged usually by the tedious process

of shovelling them over the side of the vat. A truck line runs generally by the side of the tanks to receive the discharged tailings, which are dumped outside the works. The tank is then ready for a fresh charge.

The percolation system just described may be modified however in various ways. One of the first difficulties arose when the tailings contained a small percentage of iron pyrites, which, on exposure to the air, had become converted into sulphate, with the liberation of free sulphuric acid. The trouble could be minimized by treating the tailings direct from the battery, giving no time for the oxidation of the pyrites, but to deal with old tailings which had been exposed to the atmosphere for some time it was found necessary to give them a preliminary washing, first with water and afterwards with an alkaline solution, such as lime or caustic soda. At the Robinson works lime was given the preference, as it is less active in inducing decomposition of the cyanide solution in the tanks and in attacking the zinc used in precipitation.

Substances present in the ore which are capable of decomposing the cyanide may cause much trouble by reprecipitating the gold, and depositing it in the gelatinous mass formed on the surface of the tank, especially if the circulation-method is employed. It has been stated that a ton of ore generally requires for its treatment a ton of solution, and since with free-milling ore a much smaller quantity is sufficient to dissolve the gold, it was suggested that the solution from one tank might be transferred to a second, and be made to dissolve an additional quantity before passing to the zinc boxes. For example, at the Robinson works it was found that 20 tons of solution were amply sufficient to extract 40 ounces of gold from 75 tons of tailings in one tank; whilst 20 tons of solution sufficed to fill a tank holding the usual charge of 75 tons of tailings, covering it to a depth of 3 or 4 inches.

Instead, therefore, of replacing this 20 tons of solution by fresh cyanide, the solution filtering through was continually pumped back again into the same tank for about 36 hours, and then passed through the zinc box.

The extraction of gold by this circulation-system was equal to that obtained by the ordinary method, and the consumption of cyanide was much less, since a much smaller quantity of solution was exposed to the zinc. Another modification suggested itself as a further step, viz., to transfer the solution charged with gold from one vat to a second and third in order that it might take up an additional quantity of gold from fresh tailings before going through the next stage of the process.

The advantages of this method are that the solutions from which the precipitate is obtained are much richer in gold, giving a cleaner deposit on the zinc with much less consumption of cyanide.

Whilst the usual practice is to prepare the cyanide solution in a special tank, as before stated, at the Robinson works, the strength of the liquor is kept up by adding cyanide in lumps, dissolving it under the stream from the pump. By this procedure, a cyanide solution, of required strength, is formed in the leaching-tank itself.

This simplifies the operation and diminishes the number of tanks needed, while affording an easy means of getting rid of the insoluble impurities of the cyanide (the so-called carbide of iron) which would otherwise accumulate as a black slimy deposit in the concentrated solution tank. This insoluble residue is, of course, discharged with the tailings when the tanks are emptied.

A further difficulty frequently encountered in the application of the cyanide process is the treatment of battery slimes, which, owing to their fine state of division have a tendency to agglomerate into pasty masses. These either resist the penetrating action of the cyanide or retain the dissolved gold during the leaching operation.

No satisfactory remedy had at any rate until quite recently been devised, though the evil may be lessened by mixing the slimes with clean, coarse sand. In the direct treatment of ore from the battery previously alluded to, coarse gold, which is easily caught on the plates, is very slowly dissolved by cyanide, and this is probably a serious source of loss.

The experience at the Langlaagte Estate and other works in treating pyritic concentrates with or without agitation seems to confirm what has been already stated. Although an extraction of over 90 per cent. has been obtained, it appears that the consumption of cyanide is enormous.

Various improvements have been introduced in the mechanical details of construction. The large leaching-vats of the Robinson works are rapidly and conveniently discharged by a trap-door placed in the centre of the tank bottom, which is hermetically closed by a screw-fastening. The enormous underground vats of the Langlaagte Estate works are discharged by a dredge, which appears to give satisfaction.

Precipitation of the Gold.—Having obtained a solution of gold in cyanide of potassium, the next step is to recover it. Various precipitants have been suggested, but the only one which has come into use on a large scale is metallic zinc in the form of freshly-turned shavings ; zinc in sheets offer too little surface for the deposition. The same is true of granulated zinc. When once the surface has become coated with an extremely fine layer of gold the action ceases, or becomes so slow that the precipitant cannot be practically applied in this form. Zinc dust and zinc amalgam have also been tried and are effective, in so far as they

present a large surface for deposition, but are found to clog if the continued flow of liquid through them be interrupted. Sodium or potassium amalgam has been used with success on a small scale, as in the Molloy process now about to be tried with a 500 ton plant at the works of the Pioneer Gold Mining Company, but the difficulty so far has been to manufacture these substances cheaply and in sufficient quantity on the spot.

The zinc shavings now in use are prepared by turning thin sheets of zinc on the lathe. This produces a light spongy mass, which readily allows the solution to filter through and presents a large surface for the precipitation of the gold. These shavings are placed in wooden troughs, commonly known as zinc boxes, and the solution from the leaching-vats is allowed to flow slowly through them, depositing the gold as a finely divided black slime on the surface of the zinc, while the zinc gradually dissolves in the liquid. After passing the zinc box the exhausted solution, which should not contain more than $\frac{1}{2}$ dwt. of gold per ton, flows into the storage tank or sump, whence it may be pumped back to the leaching-tanks, when a fresh charge has to be treated.

The simple replacement of gold by zinc is not the only reaction which occurs in the zinc box: we find that a notable falling off in the strength of the cyanide occurs, due to secondary reactions caused by the gold-zinc couple.

Various slight modifications have been introduced in the construction of the zinc boxes. They are usually divided into several compartments so arranged that the liquid flows alternately upward and downward through the shavings. The shavings are placed in a tray, the bottom of which is an iron-wire screen of about 4 holes to the inch. This is supported a few inches from the bottom of the zinc box. The fine gold slimes fall through this screen, and may thus be separated from the undecomposed zinc when the clean-up takes place. The zinc boxes used at the Robinson works are about 20 feet long, 2 feet wide, and 2 feet deep, with inclined bottoms. They are divided into compartments of about 20 inches in length. Each compartment holds about a bushel of shavings, weighing about 40 lbs.

Seven compartments in each zinc box are filled with shavings; a single compartment at the head is left empty to receive any sand that may be carried through the filters by the solution from the tanks. A double compartment at the foot is also left empty to allow any gold that may be carried away by the stream of liquid to deposit before the solution flows into the sump. About 60 tons of solution, which is the quantity required for treating the ordinary daily charge of 225 tons of tailings, are allowed to run off through two zinc boxes in about 9 hours. This solution may

carry from 1 to 3 ounces of gold per ton of liquid ; after passing through the zinc boxes it rarely contains more than 2 dwts., and should not contain more than $\frac{1}{2}$ dwt. if the precipitation has been properly carried out.

There are two sets of zinc boxes, one to receive the strong solutions (0·6 per cent. to 0·8 per cent. cyanide), and one for the weak solutions (0·2 per cent. to 0·4 per cent.). The slimes formed in the weak boxes are, as a rule, much poorer than those in the strong boxes, and less consumption of zinc takes place in them.

The total amount of zinc consumed amounts to about 100 lbs. per day. Two men are constantly employed at the lathes, so that the turning is an arduous and somewhat costly operation. It is desirable to use freshly-turned zinc, as the surface rapidly oxidizes and then becomes much less active in precipitating the gold. The most vigorous action of course takes place in the compartments which first receive the solution from the tanks. It is here that the zinc dissolves most rapidly, and is accordingly replaced by shavings from the lower compartments, whilst fresh zinc is continually added as the last compartment is emptied.

The clean-up takes place once or twice a month. The screens containing the undissolved shavings are lifted from the zinc boxes. The boxes are then left undisturbed for an hour or so to allow the zinc gold slime to settle at the bottom. The liquid is then drawn off by a syphon until very little is left above the slimes. The box is then cleaned out, and the slimes and muddy water allowed to drain through a screen of 40 meshes to the inch. The mass consisting of water, finely divided gold, and very fine zinc is rubbed through this screen with a short stick 5 or 6 inches long, to the end of which a piece of india-rubber is fixed.

The stuff remaining on the screen consists almost entirely of unconsumed zinc fine enough to pass through a screen of 12 meshes to the linear inch. This is replaced in the first division of the zinc boxes over a fresh lot of shavings. The slime consisting of finely divided gold and silver, with a large proportion of zinc and lead, and a certain quantity of tin, antimony, organic matter, and other accidental impurities, is allowed to settle in a small tank placed beneath the 40 mesh screen, and is now ready to undergo the drying and melting operations necessary to convert it into bullion.

In the Molloy process the use of zinc is dispensed with altogether. The solution passes through a shallow trough containing mercury, in which is an inner cylindrical vessel filled with solution of carbonate of soda, the edges of the cylinder just dipping beneath the mercury so that its contents are entirely cut off from the other portion of the vessel.

A rod of lead dips into the soda solution, the lead and mercury are connected with opposite poles of the dynamo and the solution is electrolysed by the passage of the current. The sodium released combines with the mercury to form sodium amalgam, which decomposes the gold cyanide solution with formation of gold amalgam, sodium cyanide being simultaneously produced.

It is claimed that much less decomposition of the cyanide takes place than with zinc, and that the outflowing solution is better adapted for dissolving fresh quantities of gold, becoming regenerated. In the ordinary method a large accumulation of zinc in the solutions must take place, which in time would render them valueless for gold extraction, whereas sodium cyanide is just as effective as the potassium compound. If this method of precipitation should prove successful on a large scale, a great improvement will have been effected.

Production of Bullion.—The third stage of the cyanide process consists in converting the precipitated gold into a saleable form. The slimes are transferred to enamelled iron pans and carefully dried over a small furnace. This is a tedious operation which requires time. The richness of the dried slimes will depend on the percentage of gold present in the cyanide liquors passed through the zinc boxes.

The pans in use at the Robinson works contain about 5 or 6 gallons of dried precipitate. This may contain as much as 150 or as little as 20 ounces of gold. The precipitate when nearly dry is mixed with sand, borax, and bicarbonate of soda, and melted in a No. 60 crucible at a fairly high temperature. The material melts very easily, forming a very liquid slag which, however, corrodes the pots rapidly, so that a good crucible rarely lasts out more than eight meltings. The charge is not added all at once, but as each portion melts and sinks down fresh quantities of the mixture are added.

When the pot is full of liquid slag it may contain from 100 to 150 ounces of bullion. Large quantities of oxide of zinc are given off during the melting, which carry off a very appreciable amount of gold. The zinc fumes, together with the products formed by the decomposition of the cyanide salts, render the operation anything but healthy. The bullion produced is whitish in appearance and about 650 fine. It is very hard and brittle, and the bars are by no means uniform, so that it is difficult to obtain an accurate assay. In addition to zinc they contain silver, lead, and sometimes a little copper. Several ways have been suggested for obtaining a purer bullion. One method consists in partially roasting the slimes in a muffle furnace, whereby part of the zinc is oxidized and

volatilized, leaving a much smaller mass for subsequent melting. By this treatment bullion of 800 fine may be obtained.

Another method is to carefully wash out the soluble cyanide salts and then treat them with dilute sulphuric acid, which dissolves the zinc. The objection to this process is that the slimy mass is very difficult to filter, and retains the soluble zinc salts even after prolonged washing.

The same trouble occurs when the zinc is dissolved in hydrochloric acid. Some difficulty is also experienced in washing out the soluble cyanide salts. It is possible that the use of filter-presses might to some extent solve the difficulty of purifying the zinc slimes. The use of acid sodium sulphate also appears to promise good results as a solvent for zinc.

No attempt as a rule is made to refine the product, because to melt straight away into bars of bullion yields a larger number of ounces per month, though it does not augment the sterling value of the product.

In general it is found that on a large scale the extraction of gold by cyanide amounts to 70 or 80 per cent., and tailings assaying 8 to 10 dwts. will give residues assaying 2 to $2\frac{1}{2}$ dwts.

A much higher extraction is obtained on a small scale with the same strength of solution when, relatively, large quantities of solution are allowed to pass through the material under treatment.

That the cyanide process is well adapted at present for the treatment of Witwatersrandt ores is shown by the fact that, although the system has been in use little more than two years, 40,000 tons of tailings are now being treated per month. The process is only in its infancy, the varied and complex problems to which it has given rise are mostly unsolved, its limits are yet undefined, and it would be rash indeed to forecast its future.

Turning again to what has been done in America, some interesting particulars of the process are given by Mr. C. Merrill in the *New York Mining Journal* of November 5th, 1892, from a report made by Messrs. Louis Jannin, Sen., and Henry Bratnober on the results of the process at the Mercur gold-mine, Fairfield, Utah. The ore is a siliceous limestone, carrying magnetic oxide of iron, traces of cinnabar, and gold (no silver). No free-gold appears to be present—in the ordinary acceptation of the term—though the magnetic oxide of iron appears to be more or less coated with a thin film of gold. The ore contains considerable silt, and any attempt at fine-crushing results in sliming the greater portion. This being fatal to successful leaching, coarse-crushing had to be resorted to, which has the disadvantage of involving more time for successful treatment. There appear in fact to be few ores which can be treated with any

degree of success unless crushed fine enough to pass a 20 or 30 mesh screen. The ore after crushing in a stone breaker passes through two sets of corrugated rolls, which give a product, 20 per cent. of which remains on a No. 4 screen, 40 per cent. on a No. 12, 13 per cent. on a No. 30, and 26 per cent. passes through. Of this last 26 per cent. nearly the whole is impalpable powder.

The ore is carried by an overhead tramway on cars to the leaching-vats. The best size and pattern for the latter, according to experience at this mill, is a round vat, the shell of which is No. 10 or 12 sheet-iron. The bottom is of 3 inch California red wood, and is caulked with oakum, over which is poured a mixture of tallow and resin. On this bottom are placed 1 inch by 1 inch slabs 18 inches apart. Upon this rests a false-bottom of 1 inch yellow pine, in every square inch of which is perforated a $\frac{3}{4}$ inch auger hole. Over this perforated false-bottom is stretched a burlap filter. There is, of course, a stopcock (between the true and false-bottom), which should be of iron. The dimensions of the vat are: diameter, 12 feet 8 inches; depth over all, 40 inches; depth to false-bottom, 35 inches; capacity, 14 tons. There seems no reason, however, why larger vats should not be used, provided they are round.

The ore having been charged, is levelled to within about 6 inches of the top of the vat, and the stopcock being closed, a $\frac{1}{4}$ per cent. solution (1 lb. of pure potassium cyanide to 400 lbs. of water) from the standardizing-tank, is run in from a pipe till about 3 inches of solution covers the top of the ore. It has been found that the solution acts slowly at first, but more rapidly after extraction has begun, possibly owing to some galvanic action.

The charge is allowed to soak from 12 to 24 hours, then the solution is allowed to percolate, flowing in at the top and out at the bottom for from 24 to 240 hours, according to the leaching rate of the ore. That from near the surface of the mine is very slimy and requires a longer time. The average time is about 60 hours, or practically until the outgoing solution does not discolour bright zinc. The test is made by placing a little sieve containing bright zinc threads under the stopcock, and allowing it to remain there for an hour or so.

If it remains bright the solution has extracted all that it will extract, and the flow is stopped. The outgoing solution from all the vats flows to a sump, and is pumped thence to a second or gold-solution tank. From this tank it is allowed to flow constantly through two boxes containing spongy or thread zinc, each box being 40 feet long, one of wood 12 inches square internally, and the other of iron 15 inches square. Each box is provided

with partial partitions which deflect the current from the bottom to the top and *vice versa*. These partitions are placed about 3 feet apart. The zinc should occupy so many of them as will give bright zinc in the last one, so as to ensure complete precipitation. The solution flows from the zinc boxes back to the standardizing-tank, where it is occasionally tested, and if below the required strength, potassium cyanide is added in proper quantities.

After the gold has been extracted and the solution has been turned off, the vats are allowed to drain. It has been found that there remains 400 lbs. of $\frac{1}{4}$ per cent. cyanide to the ton in the tailings, and to force this out a wash of about the same quantity (400 lbs.) of water is used, which joins the main body of rich solution in the sump.

After this is drained out, as the tailings still contain 0·3 to 0·4 of a pound of cyanide per ton, a second wash of 400 lbs. of water to the ton is added, and the very weak solution left (which is forced out), runs to the waste-solution tank, whence the first wash of succeeding charges is drawn.

At the end of the month the outlet from the gold-solution tanks to the zinc boxes is closed and the latter are allowed to drain. When comparatively free from solution the richest portion of the zinc product, which has the property of powdering up in the fingers, is removed.

Mr. A. Hanauer gives the following analysis of this zinc and gold product:—Zinc, 39·1; carbonate of lime, 36·7; gold, 4·4; cyanogen, 3·5; sulphur, 2·6; iron, 2·4; and residue, 6·0. This product is sampled and treated by the Omaha Smelting Company, who return 20·60 dollars per ounce of their assay of its gold contents. Deducting 12 cents for express charges, this leaves a net return of 19·88 dollars per ounce of gold extracted.

It is stated that from April 1st to July 1st, there were milled 1,513 tons of moist ore of an average value per dry ton by assay of £3 3s. 5d. (15·22 dollars). The ore tailings' assay was 10s. 10d. (2·60 dollars). The apparent extraction was therefore £2 12s. 7d. (12·62 dollars) per dry ton, or 83 per cent.; hence, without allowing for moisture, we should have a bullion return of 19,272·60 dollars. The actual bullion returns from the smelter were 16,805·80 dollars = 73 per cent. = £2 6s. 0d. (11·04 dollars) per ton of moist ore. The discrepancy is accounted for by the ore being weighed wet and assayed dry, making a difference of at least 6 per cent. due to moisture, besides loss by leakage, handling, and drying.

The losses due to leakage and handling will probably be minimized when the mill is in thorough working order, and should, it is claimed, leave a difference of not more than 2 to 3 per cent. The following is

stated to be the itemized cost per ton, compiled from the books of the company. It is exclusive of superintendence, office expenses, and royalty, and covers a period of six months from January 1st to July 1st :—

	s.	d.
Potassium cyanide, 1·27 lbs. per ton	2	9
Zinc, 0·55 lb. per ton	0	2½
Labour (7 shifts per 24 hours, 6 day and 1 night) ...	4	8
Supplies, repairs, fuel, freight, etc.	2	4½
	10	0

In regard to the labour item it is said that since the period covered by these figures the capacity of the mill has been doubled, reducing this charge to 2s. 4d. per ton, and the total cost to 7s. 8d. per ton. It is intended in the near future to again double the capacity of the works, requiring only the addition of 4 shifts to run the crushing machinery and to charge the tanks at night. At present only the solution-man is on night shift. Notwithstanding the difficulty the Mercur ore offers to leaching, (owing to its disposition to slime), the process appears to be in this case certainly an economic success, but it cannot be repeated too strongly that extensive preliminary investigations should be made by an impartial person before attempting to apply it to an untried ore in a new locality.

Raw Grinding.

As to the amalgamation of gold ores, Messrs. McDermott and Duffield remark :* "Generally the pan-amalgamation process must be preceded by roasting. In the colonies, grinding and working in pans the raw, partially concentrated sulphides is practised, but in the United States this process has entirely gone out of use, being displaced by smelting and chlorination, rendered possible by the development of the country and more perfect concentration."

Some sulphides contain their value in comparatively coarse-gold, which, grinding will liberate and mercury attack, but most of the base-metal sulphides in gold ores will only yield a small part of their gold contents to such a process. Even when still in use this process has its chief excuse in an imperfect concentration, and would be better superseded by the close saving of higher-grade clean concentrates, properly treated by chlorination, or some modification of the smelting process.

The same writers remark :† "The sulphide ores and concentrates can, as a safe general rule, be considered as not adapted to amalgamation without a previous roasting. In some few cases where the gold is comparatively coarse, a sufficient proportion of it can be extracted by

* *Gold Amalgamation*, page 31. † *Ibid.*, page 44.

raw pan-amalgamation to pay a profit on the operation, and this process is used at places in the Australian colonies, usually after a crude process of concentration, which itself involves considerable losses of fine minerals, and the production of unclean concentrates, and in some few cases the value thus extracted is to an appreciable extent in the form of amalgam lost before concentration. Generally speaking, therefore, it is safe to say that raw amalgamation is inadvisable on the material now treated of.

"The dead (oxidizing) roasting of sulphides as a preparation for pan-amalgamation improves the process by preventing the loss of quicksilver consequent on working raw sulphides, but it is not so successful as is generally supposed, or as theory would promise. The operation of roasting (whilst it apparently frees the gold from its combinations, increases, perhaps, the size of the gold particles, and eliminates the objectionable sulphur compounds,) seems to have a bad effect on a large part of the gold, putting it superficially at least in a condition unfavourable for securing contact with quicksilver. The extent to which the gold is thus affected varies curiously in different ores of apparently similar composition, and the conditions of roasting have also an influence on the result.

"For the amalgamation itself of the roasted ores various methods are recommended. One process used in the colonies is by working with large excess of quicksilver, and little water, and apart from contact with iron. The effect of grinding in iron pans seems in some cases much less beneficial than when amalgamation is conducted in stone arrastras. The use of gold amalgam in place of quicksilver, and the avoiding of contact with iron surfaces, was found most beneficial in experiments conducted by Mr. Stetefeldt, in Mexico, and a high percentage was extracted from a low grade ore.

"On the other hand, experiments on many gold ores in the New York ore-testing works gave unsatisfactory results by all these processes after roasting in a small reverberatory furnace, and nothing but chlorination was found effective when the material was rich in gold."

On most points the writer is perfectly in accord with the views that have just been stated in regard to raw amalgamation, and entirely so in regard to the inexpediency of roasting first, if only because of the extra cost added to the process when the ore is roasted, which scarcely any ore can bear, but he believes that, in certain exceptional instances, as regards the ore and as affected by locality, the process of raw-grinding is justifiable to a certain extent.

The writer thinks, as Dr. Foster pointed out in the discussion on Mr. Curtis' paper, that it is perfectly possible to extract a very high

percentage of gold from heavy pyritic ores and concentrates by this method, of which there are examples in the Pestarena ore in Italy (amalgamating it in Frankfort mills) and in some of the Transylvanian gold mines, where Tyrolean mills are employed for a similar purpose.

More frequently, however, one finds where an ore carries heavy pyrites it is not what one would term free-milling, and only a certain proportion, roughly speaking 50 to 65 per cent., can be got out by raw grinding, the process being generally accompanied by heavy loss of mercury and high working costs.

It is still more often the case, that mere grinding without previous natural decomposition (which can sometimes be assisted by adding a little salt to the ore pile and allowing it to weather for two or three months previous to treatment) will only extract a very insignificant percentage indeed, and roasting with salt generally will merely tend to make matters worse, both commercially and metallurgically.

The method of grinding, just alluded to, which is followed at Pestarena, in Italy, is in several respects unique, and cannot be passed over without special comment.

The ore, which consists of quartz, mixed with micaceous, graphite-schists, carrying iron pyrites, associated with a little arsenical pyrites, is first screened, and the lump ore, after being picked over, and if necessary cobbed, is crushed in a rock-breaker, and then delivered to Cornish rolls, whilst the screenings (mine fines) are trammed direct to the rolls, unless of too poor quality, in which case they are concentrated first by jigging.

The medium-size ore (resulting from cobbing) is washed and carefully picked over on tables, provided with settling-tanks to catch any fine pyrites that might otherwise escape, before it is sent to the stone-breaker.

The product of the rolls derived from these different sources is delivered to bins, connected with what are known as Frankfort mills, an improved form of arrastra.* 5 to 10 lbs. of lime per ton of ore is added to the charge to prevent sickening of the mercury. The action of sickening may be caused by greasy substances such as graphite, which, under the name of kish, is often used to cover the surface of castings, oil, grease, etc. Metallic oxides and finely divided carbonate of lead likewise cause flowing, and so do some soluble chlorides, owing to the formation of calomel on the surface of the mercury. Ferric-chloride being formed when ores are amalgamated in an iron pan with salt and sulphate of copper, will account

* The writer intends to make the construction and operation of these mills (for a description of which he is indebted to the kindness of Messrs. John Taylor & Sons) the subject of a separate paper, he will therefore pass over their description here, though of very great interest, without further comment.

for this action in some cases; where it is due to metallic chlorides a little metallic zinc added to the quicksilver will preserve it; when caused by graphite, which has been used to coat a mercury trough for instance, the only remedy is to carefully varnish it with shellac.

The quantity of ore treated in each mill varies from 10 cwts. to 1 ton per 24 hours, depending on the proportion of pyrites that it carries.

The report of the Pestarena company, for the year 1888-89, showed that 4,474·811 tons of wet ore, representing 4,345·443 dry metric tons, or 4,276·785 dry English tons, were milled in this way, at Val Toppa, during the 12 months ending June, 1889, with 20·5 mills, working 309 days. The results showed an extraction in bullion of 8 dwts. 23¾ grains, per English ton of ore. The bullion being 797·1 fine in gold, and 197·8 fine in silver. The fine gold, determined in the crude ore by assay, amounted to 8 dwts. 20 grains, per English ton, of which 7 dwts. 4 grains, were extracted by the mills, equivalent to a duty of 81·1 per cent., with a loss of 234 grammes of mercury per metric ton of ore treated.

The report of the same company for the year ending 1889-90 showed that 5,916·771 metric tons of wet ore, representing 5,724·004 dry metric tons, or 5,633·560 dry English tons, were milled at Pestarena (a separate establishment) during the 12 months ending June, 1890, with 25 9 mills, working 344 days. The results showed an extraction in bullion of 18 dwts. 8 grains per English ton of ore, the bullion being 742·4 fine in gold, and 252·2 fine in silver. The fine gold determined in the crude ore by assay, amounted to 17 dwts. 8⅓ grains per English ton, of which 13 dwts. 15¼ grains was extracted by the mills, equivalent to a duty of 78·77 per cent., with a loss of 230 grammes of mercury per metric ton.

Looking at these results, the question arises as to how far the extraction in grinding is affected by the fineness of the metal present entering into the bullion, the degree of concentration, and the grade of the ore, and it seems to lend confirmation to a point observed in other cases, that a low-grade ore, carrying fine metal (gold or silver) with a large percentage of gangue or sand in the concentrates, yielded in proportion a higher return than when these conditions are found reversed.

It may also happen that the loss of mercury is affected by the duty of the mills. The agents, Messrs. John Taylor & Sons, in their report on the Pestarena mines, for the year 1888, stated that the duty or percentage extracted of the assay contents of gold in the ore had increased, being 0·80 as against 0·78 last year; and the consumption of mercury had increased from 242 grammes to the ton during last year to 310 grammes per ton.

If the loss of mercury were due, as might be imagined, to the extra baseness of the ore, one would expect the yield to fall off, whereas the contrary is the case. The writer is inclined, therefore, to think that it may be rather attributed to extra grinding of the ore, which would tend to increase both the yield of gold and loss of quicksilver. Professor Le Neve Foster has remarked that on the whole more gold is extracted in these works in winter, when the water, coming from the mountains, is comparatively cold and clear, than when it is turbid and warm in summer. In July, 1887, for instance, at Val Toppa, the duty of the mills was 70·6 per cent., and varied from September to December from 91·7 to 88·5 per cent. Though it does not, in this respect, appear to follow an invariable rule in different years, this may be accounted for by modifying causes in different seasons, character of the ore and its general manipulation, as to amount of grinding, etc., and perhaps, as Dr. Foster thinks, the turbidity of the water, as well as the more rapid oxidation of the pyrites in summer, may partly account for these results.

The conclusion he draws that low temperature is not incompatible with good amalgamation in this instance, is no doubt perfectly true as well, since a certain amount of heat is generated by the mechanical action of grinding;* but of course this statement is not intended to apply to plate-amalgamation.

In Dakota, for example, where the thermometer sinks, at times, to 40 or 50 degs. below zero, provision is always made for heating the supply served to the stamps by passing the waste-steam of the engine through a steam-coil in the supply-tanks, and there is generally a steam-drum as well, running in front of the apron-plates, as an extra precaution.

Either very cold or warm water† would certainly be pernicious in plate-amalgamation. Speaking of the writer's own experience of grinding raw sulphurets in pans, he made some careful experiments in 1884, treating a quartzose ore, carrying on an average $2\frac{1}{2}$ per cent. of pyrites, which was concentrated on end-bump tables. He put a lot (about 14 tons) through iron wheeler-pans, subjecting them to continuous amalgamation for 8 hours on somewhat the same plan as the American Boss system (so called after its inventor Mr. H. P. Boss), and found that though the concentrates originally assayed only 1 oz. 16 dwts. 18 grains, the tailings still contained 17 dwts. 12 grains per ton.

* Dr. Foster roughly measured the increase in temperature in course of grinding in a Frankfort mill at Pestarena, and found that an average rise of about 3° Fahr. took place, but on the mill-bed itself the heat must have been much greater.

† The-pan amalgamation of silver ores, it is true, is facilitated by a high temperature (live steam being often introduced to warm the pulp), but the conditions in that case are entirely different.

In the Black Hills, it has long been the fashion to treat blanket concentrates in pans. Dr. Hofman states* that when panned down for experiment, these gave 20·5 per cent. cleaner concentrates, assaying 40·18 dollars per ton, but when amalgamated in a wheeler-pan, only yielded 56·9 per cent. of their gold contents.

In Australia, again, the writer has constantly found that battery-pulp concentrated on percussion-tables and buddles, up to 60 and 75 per cent. of pyrites, when ground in berdan-pans, only yielded 50 to 60 per cent. of its gold in the first grinding, though about half as much more might be extracted by a repetition of the process at much extra time and cost.

The Warden in his official report on the Charters Towers gold-field for 1878, page 14, remarks in connexion with this subject: "There can be no doubt that in many instances more gold escapes with the tailings than is in the first instance obtained by the crushing mills. For support of this, I am aware of several hundred tons of tailings having been put through the stamper boxes as a trial, and the result obtained was 15 dwts. to the ton. If such a yield is obtainable in such a manner, I may well ask the question, what must be the yield of tailings when properly treated with the most approved gold-saving appliances? Practical men of long experience inform me that tailings† on this field, under proper treatment, will yield on the average 9 ozs. 5 dwts. per ton." The writer does not think that the average loss of gold at the present time in the pan-tailings of Charters Towers is much less, however, than 14 dwts. to the ton.

The only remedy that has been applied to any large extent is grinding and re-grinding. The re-grinding of these tailings (after the miner has relinquished his interest in them), was at one time a regular recognized business, and is doubtless still carried on to some extent at some of the mills.

This, no doubt, accounts for no small portion of the gold, indirectly supposed, if not alleged to be stolen by the miners working in some of the mines, or by the mill hands, an explanation the writer does not for one moment believe, though it may certainly be a convenient red herring to draw across the scent to divert investigation from channels it might possibly follow with more advantage to the general mining community and the industry at large.

There may be small isolated thefts of specimens, but, taken as a body, the writer has always found the Queensland miners an uncommonly honest set of men, and it is quite impossible that 18,593 ozs. to 15,169

* "Gold-milling in the Black Hills," by H. O. Hofman, *Trans. Am. Inst. Min. Eng.*, vol. xvii., page 538.

† The allusion here is evidently to pan-sludge after concentration and grinding.

ozs. (the excess of gold purchased by the banks over the mill returns in 1889 and 1891*) could have been illicitly obtained and secretly crushed.

As the mills do not publish any record of the gold got from the source referred to (that the writer is aware of), there is no check on the matter, and the miner who has an interest in a mine or claim (and there are few who have not) may more probably be the real sufferer,† whilst being branded actually as the culprit.

In pan-amalgamation, the flowering of the mercury which Mr. Curtis has alluded to, even when heavy pyritic concentrates are being worked, can generally be avoided with care, by selecting a suitable class of pan, grinding the ore to sludge before adding the mercury, and raising the shoes (if wheelers are used) off the bottom, during amalgamation (so as to allow the muller merely to act as an incorporator), keeping the pulp of the proper consistency (neither too thick nor too thin) and using chemicals judiciously, adapted to the particular conditions that obtain.

If the pulp gets too thick, the inevitable loss of mercury in treating ore in berdans is likely to be greatly intensified; using wheelers, the charge should be about the consistency of cream, and to avoid loss, the settler must be properly looked after.

A consideration of the MacArthur-Forrest process, also forces one to consider the losses of the precious metals, (more particularly gold in solution), that may occur through the indiscriminate use of chemicals, such as cyanide. Other re agents such as salt, iron, sulphate of copper, sal-ammoniac, and caustic soda and potash, are used by different millmen, in certain cases, in the amalgamation of gold and silver ores.

Cyanide keeps the mercury quick and lively, and the last-named reagents neutralize greasy substances introduced into the pan, for which wood-ashes are also employed. Caustic lime is sometimes used with roasted ores which contain a great deal of cupric chloride, to reduce it to cuprous chloride. Sodium amalgam is useful, especially for ores containing binoxide of manganese, to prevent flowing, which that metal, as well as copper and lead, are more especially liable to occasion, by debasing the amalgam.

Blue-stone, salt, and iron assist the decomposition of certain silver ores, and an acid solution of the former salt, when there is a large proportion of clay in the ores, which causes it to ball-up and carry off globules of mercury, probably acts on the clay like alum, tending to promote its precipitation and prevent loss.‡

* *Charters Towers Gold Mines*, by L. W. Marsland, page 197.

† *Vide* remarks on page 204.

‡ Tenth Report, *Census of the United States*, 1880.

Dr. Raymond and others have frequently pointed out that some gold is lost in solution in ordinary plate-amalgamation, but if cyanide is the active agent in attacking gold and silver which it is represented to be, it stands to reason that very large hitherto unsuspected losses may occur in certain instances, especially when the gold is naturally in a fine condition.

We do not know how large these losses may have been in the past, because the writer ventures to say that there is no mill in the world where amalgamation-tailings are regularly tested for gold in solution, but if we have not examined the question hitherto, it is one that deserves to be looked into in the future where cyanide is used to any extent. A little red-oxide of mercury dissolved in the cyanide, will often greatly assist amalgamation, and for the recovery of the dissolved gold, a little zinc amalgam may be added to the pan towards the end of the operation.

Raw-grinding is still the favourite process in North Queensland, and it is likely to remain in fashion there for many years, for good reasons, or otherwise, although it has been proved to be a more or less mistaken idea.

The very fact that it pays local chlorination-works to purchase and treat the pan-tailings after the first grinding, shows that it cannot be otherwise than sheer waste of money with some of the Charters Towers ores to incur the first expense of grinding at all. Amongst the reasons alluded to may be mentioned :—

1. The heavy loss of capital involved in making a radical change in a field where all the mills are laid out specially for grinding, as the large number of pans which are now running would only be worth the value of old iron if they were thrown out; affording a good illustration of the importance of choosing the proper kind of plant in opening up a new district, making due provision for probable changes in the class of ore likely to be met with in depth.
2. The fact that some of the Queensland ores contain considerable amounts of base metal, especially lead, and these can be treated at some profit by grinding-milling, whilst it has yet to be determined how far this class of stone can be otherwise worked to better advantage; smelting being out of the question at present.
3. The difficulty of overcoming the general prejudice on a gold-field against any new form of procedure.
4. The fact that as many claims are worked by prospectors, and as many of the mills are run on custom-work, chlorination-works would have either to work a number of small lots separately, which would be costly and impracticable, or else buy the ore outright, which introduces the next difficulty.

5. That the miner, with a wisdom born perhaps of experience, likes to follow his gold and look after it himself from start to finish, and would not be convinced that selling it on assay the advantage did not gravitate to the side of the mill. It is only when he has followed it just as far as he can, and scraped every crevice of the mill himself (to see that there is not one atom of amalgam lost), that he is ready to let the tailings go out of his own control and sight.
6. A change would perhaps temporarily throw a small number of men, accustomed to grinding-milling out of employment, and introduce the necessity of specially skilled superintendence, so that any alteration in the existing *régime* might affect local capital and local management in these respects, and as the gold-fields policy of Queensland, when controlled by local influences, has up to the present turned apparently on the exclusion of everything British except money, the change is not likely to come yet awhile.

The chief sufferers, of course, are the miners themselves (who are mostly shareholders in the mines but not in the mills,) and outside capital introduced into the district, representing at present, the nominal amount of £1,737,268 out of a total of £3,774,400, invested in mines, in the colony of Queensland. Both the miner and capitalist are, it would therefore seem, paying to maintain a wasteful system, which benefits comparatively few people, by which the country loses a large amount of gold, which would be better in circulation in the pockets of the community, than in the river bottoms.

In districts, like Charters Towers, no doubt, we shall ultimately see most of the mills adopting the principle of simply catching all the free-gold they can in the battery and on plates,* concentrating thoroughly, and disposing of the pyrites to central chlorination-works, merely utilizing the pans as they stand for treating the tailings of the concentrators after they have been classified, whilst large outside mines, which have ample stocks of pyrites, will, in time, put up chlorination-works of their own with advantage ; but smaller concerns will no doubt do best to stick to grinding and suffer the loss for the time being.

Where gold ores must be ground, it would be better, in the writer's opinion, to grind in Frankfort mills, or to substitute Chilian mills for

* A few pans might, of course, be kept for occasional use, to deal with ores that could be advantageously treated by grinding, under conditions described.

berdans, and amalgamate on copper plates outside, grade the pulp in classifiers, concentrate on efficient tables, and treat the concentrates by chlorination, or else where pans are employed for the treatment of sulphides (concentrates), to grind continuously in large-sized wheelers, with wooden sides (in place of the small iron ones that are in general use), and amalgamate afterwards with the mullers raised; or as another alternative, distribute the ground-up pan-sludge to berdans, discharging into large settlers provided with proper drags.

The writer's reason for this opinion is that even an ordinary wheeler* is a far faster grinder than a berdan, whilst the latter machine seems to be a better gold-amalgamator.

The writer ventures, however, to think that the statement that grinding in the colonies is the outcome of poor concentration, ought to be reversed, and for this reason, that a certain proportion of sand must be present to brighten the liberated gold and lighten the charge.

Another special point to observe is that a berdan grinds and amalgamates better, and naturally wears out less, if the drags are made of softer iron than the pan itself.

Hardwood drags may be used if the pan is intended simply as a machine to amalgamate and collect amalgam in. It is of course a first essential that the pan should be speeded right, and inclined at the proper angle from the vertical, depending on its diameter. The gold is amalgamated by the pan being allowed to run full for some time without overflow, water being afterwards turned on from a pipe to expel the ground-up sludge.

Eight berdans are about an ordinary allowance to a battery of five stamps (which crush on an average 2 tons per head per day) for dealing with the concentrates from the ordinary run of Queensland ore, carrying 2 to 6 per cent. of pyrites, though this allowance is often exceeded for re-grinding and treating extra heavy ores.

The cost of 32 berdans (the number we may assume on this basis that would be required) for a 20 stamp battery, including the shed to cover them, erected in running order at Charters Towers, would not be far short of £1,200 to £1,300, and the 32 pans would have a working capacity of something like 28 to 30 tons of sand and concentrates weekly, allowing, of course, for variations in the method of treatment, degree of concentration, and amount of re-grinding to be done.

* The writer alludes to the small cast-iron type in common use, not the wooden-sided combination pan used in silver-milling, between which and a berdan, in the matter of grinding, there can be no need for comparison.

The cost of this treatment may be estimated as follows :—

	Per Berdan. £	s.	d.
Wages per month	0	18	2
Wood „	0	11	10
Mercury „	1	2	9
Castings, wear of belts, etc., per month...	0	9	1
Handling sludge	0	3	3
Oil and chemicals	0	1	9
Total cost	£3	6	10

or equal to 2s. 6d. per day.

Assuming that each pan grinds about 3 cwts. per day (dry weight) of 60 per cent. concentrates, it is equivalent to saying that one pan grinds 201$\frac{3}{5}$ lbs. of clean concentrates per day, or 1 ton of pure pyrites in, say, 11 days, at an actual mill cost of £1 7s. 6d.* for a yield of 50 to 65 per cent. of the gold which the concentrates contain.

Taking the cost of the grinding plant (covered in) at £1,200, and reckoning interest on the spot at 4 per cent., we must add to the above costs £48 per annum, or 7$\frac{1}{2}$d. to 8$\frac{1}{2}$d. per ton of concentrates ground, equivalent to about 11$\frac{1}{2}$d. per ton of pyrites, making a total grinding cost of £1 8s. 5$\frac{1}{2}$d.† per ton of clean pyrites.

Looked at in another light, the actual working cost of grinding the Charters Towers concentrates in berdans may be reckoned at 2s. per crude ton, added to the ordinary milling-costs, in a 20 stamp battery with a duty as stated.

The Northern Miner of February 14th, 1889, gives an interesting account of a trial made at Charters Towers by Mr. Millet, a well-known miner, at the Mary Louisa mill, on an unusually heavy pyritic ore, from the 1 and 2 West Wellington mine. Eight tons were crushed as an experiment and yielded 4 tons 4$\frac{1}{2}$ cwts. of concentrates, 2 tons 4$\frac{1}{2}$ cwts. of which were taken for trial. This was distributed between the wheelers and berdans in the proportion of 1 ton 2 cwts. treated in the former, and 1 ton 2$\frac{1}{2}$ cwts. in the latter. The wheelers gave 2 ozs. 3 dwts., the berdans 2 ozs. 6 dwts. of gold. The berdan put through one wet ton per week, at a mill charge of £2, while the wheeler put through 5 to 6 tons, at a charge of £9, or about the rate of £1 12s. 9d. per ton, leaving a margin on charges in favour of the wheelers of 7s. 3d. per ton. The total extraction by milling and grinding‡ was 31 ozs. 14 dwts. from the 8 tons of stone treated, not counting the gold left in the pan-sludge, or the

* These figures are, of course, liable to fluctuate somewhat (up as well as down), depending on the relative percentage of sand, sulphides in the concentrates, and the actual percentage of the latter in the crude ore. † *Ibid.*

‡ Even with docile ores this seldom reaches more than 76 per cent. of the total gold in the stone at Charters Towers.

2 tons of concentrates left unground. If these latter had been ground the yield would doubtless have been increased by another 4 ozs., or an equivalent of about 4½ ozs. per ton.

As over 55 per cent of the total gold was got by amalgamation in the battery it would be extremely interesting to know what the pan sludge contained after grinding.

The extraction, it will be noticed, per ton of concentrates, was 2 ozs. 0 dwt. 20 grains with the berdan, as compared with 1 oz. 19 dwts. 4 grains with the wheeler, or a difference of 1⅔ dwts. in favour of the berdan, and valuing this gold at 3s. 5d. per dwt. it really represents a saving of 5s. 8d. per ton, off-setting the smaller mill charge employing the other pans.

Now, if we assume the above saving to be fairly representative (as regards the two methods of treatment), estimating the cost of operating the berdan at 3s. 5d. per ton in excess of the cost of the wheeler (an amount seldom exceeded in a 20 stamp mill), it will be apparent that the advantage to the mill owner, crushing stone of his own, under existing conditions, would rest on the side of using the berdan, as he would gain the difference between 3s. 5d. and 5s. 8d., netting a profit of 2s. 3d. per ton. As there are chlorination-works already established at Charters Towers we can further compare the cost of grinding with that of chlorination in this same district.

Mr. Brown, in a letter published in *The Northern Miner*, October 23rd, 1886, gives the cost of vat-chlorination as carried on by the North Queensland Pyrites Co., at the Burdekin works (of which he was formerly manager), as follows :—

Cost of roasting 15 tons of ore at Charters Towers in an ordinary reverberatory furnace :—

	£	s.	d.
Labour, 6 men	18	18	0
Firewood	7	16	0
Water	0	10	0
Salt	1	0	0
Kerosene	0	7	0
Total cost	£28	11	0

or £1 18s. 0d. per ton.

Cost of chemicals and labour chloridizing 26 tons of ore :—

	£	s.	d.
Management	6	0	0
Labour, 2 men	6	0	0
Water	1	0	0
Salt, 3 cwts. at 6s.	0	18	0
Manganese, 2¼ cwts. at 5s. 6d.	0	13	9
Sulphuric acid, 416 lbs. at 5s. 6d. ...	5	4	0
Sulphate of iron	1	10	0
Kerosene	0	5	0
Total cost	£21	10	9

or 16s. 7d. per ton ; making the total cost, £2 14s. 7d. per ton.

The capacity of the furnace represented about the weekly capacity of the works, viz., 15 tons, and in the period ending September 1st, 1886, they paid 2s. in dividends* upon 740 tons of ore treated; £450 additional was invested out of profit in new plant, whilst the total capital cost of the works, up to the date in question, appears to have amounted to only £2,830 5s. The returns from the ore (pan-sludge) treated was 1,466 ozs. 8 dwts., or over 2 ozs. per ton. Owing to the success this trial plant had achieved it was proposed in 1886 to quadruple its capacity at an outlay estimated at between £4,000 and £5,000 additional, and Mr. Brown, in the letter previously alluded to, estimated that with an improved roasting-furnace† the time was not far distant when he would be able to treat a ton of ore at a cost of £1, working on a larger scale.

Mr. Brown goes on to say: "This field is exceptionally favourable for the establishment of extensive chlorination-works owing to there being a large percentage of float-gold, which is unfavourable for amalgamation, but is easily and perfectly extracted with chlorine. The time is close at hand when chlorine will do away with such extreme amalgamation as is now in practice, as the gold will be obtained much cheaper with that agent. Concentration is a matter which is much overlooked at present, and there is a large percentage of fine mundic passing off with the quartz-sand that could be easily saved by improved machinery, but this will not receive much attention until there is a demand for the concentrates, when it will be looked to. The large heaps of quartz-tailings, now looked upon as worthless, will be reduced to one-half their present value, and will pay handsomely under the chlorine process."

Now, if we assume on a liberal estimate that 80 per cent. of the gold in the pyrites can be extracted by a double grinding, and on a low estimate 85 to 95 per cent. by chlorination, we find that:—

	Per Ton. £ s. d.		Per Ton. £ s. d.
The cost of chlorination per ton of clean concentrates (page 207) amounts to	2 14 7	to	1 0 0
Add to this interest on a capital outlay of £2,800 to £7,800 at 4 per cent. on a capacity of 740 and 2,960 tons treated per annum	0 3 0¼	„	0 2 1¼
The total cost of chlorination is from	£2 17 7¼	„	1 2 1¼

Now, the cost of grinding per ton of clean concentrates has been shown to amount to £1 8s. 5½d. Therefore doubling this sum for re-grinding,

* On a nominal capital of £10,000, representing a net profit of £1,000 in eleven months, the year commencing August 12th, 1885, and ending September 1st, 1886.

† Which would reduce the cost of roasting to 10s.

the cost of obtaining 80 per cent. of the gold by this process, under the most favourable circumstances, will not be less than £2 16s. 11d., as compared with a yield of, say, 90 per cent. (to put it low) at a cost of £2 17s. 7¼d., under the most adverse conditions of chlorination.

If, in fact, we assume clean concentrates to run only 2 ozs. to the ton, the difference merely in yield at the average price of Charters Towers gold, will represent a money value of 13s. 8d. saved by chlorination, or a net gain of (13s. 8d. — 8¼d. =) 12s. 11¾d. per ton. With higher grade concentrates, and the costs reduced even to £1 10s. (as they easily ought to be), the gain with chlorination would be simply enormous. The question whether it is expedient or not to incur large gross cost in treatment for the sake of close extraction depends, in fact, not only on the quantities to be dealt with, but likewise on the grade of the material treated, it is evident that the richer this is the more important it becomes to save close.

With a single grinding, the results are scarcely less disadvantageous to the pan process, as the mine owner loses the difference between 65 per cent. and 90 per cent. of the gold in the pyrites, or, say, 10 dwts. per ton (worth £1 14s. 2d.), dealing with 2 ozs. sulphide ore.

The Charters Towers pyrites works, situated at Charters Towers itself, employ the vat process in the same way as the Burdekin works. The only peculiarities with regard to them are that they manufacture their own sulphuric acid on the spot, and employ a unique form of roasting-furnace—an idea which originated with Mr. D. A. Brown (who has been already alluded to), their able and energetic manager.

Mr. Brown conceived the plan of building his furnace on the side of a hill, possessing approximately the natural slopes requisite for the purpose he had in view, viz., to economize labour in manipulation and save as much fuel as possible (by utilizing the combustion of the sulphur in the ore, for roasting) without sacrificing the sweetness of the roast.

The furnace is shaped like a Fortschaufelung, with a number of doors on each side throughout its length.

To break up any lumps in the material (pan-sludge) to be roasted as it comes to the charging-floor it is first passed through a Carr disintegrator, driven by a pulley, provided with a coned friction-clutch. Below the disintegrator a hopper discharges the ore into three vertical circular openings, at the head of the hearth, each fitted with an Archimedean screw, driven by bevel-gearing. The speed of these screws regulates the feed, and delivers the ore into channels, which run down the upper part of the hearth.

Speaking from memory (as the writer has mislaid some notes he made when he visited the works in 1889), the hearth is about 600 feet in length, and has three different grades, commencing with 35 degs. at the head of the furnace, for a few feet, continued on for some 20 feet at a grade of 27 degs., and for the remainder of its length sloping at 18 degs. The fireplace and wrinkle (with working-openings on each side), back-up, and form the foot of the furnace.

Just below the brickwork of the hearth, three parallel lines of 6 inches fireproof stoneware pipes are laid the greater part of its length, connected at the two ends by bends, so that the artificial draught produced by a Roots blower, at the head of the furnace, circulates backwards and forwards three times through the pipe till it is finally discharged in a superheated condition, into the fireplace. The downpipe of the blower is provided with a gate-valve to prevent the furnace gases being drawn back into it when the blower stops.

The roof of the furnace is very low and flat, but presents a succession of 6 feet to 8 feet span transverse arches, with their curtains set crossways to the length of the hearth (which is about as wide as that of an ordinary reverberatory furnace) to throw the flame down on to the ore.

It was a bold project, which deserved success, though at first sight it would appear to be beset with many practical difficulties, in controlling the ore during its descent, so as to obtain a dead sweet roast during its gradual passage, from the top to the bottom of the furnace ; and at the same time to avoid large losses in dust, not to mention difficulty, from the superheated gases entering the lead-chambers.

Still, from the fact that the furnace has been in active operation almost continuously from the time it was properly started, these objections would seem to have been overcome. Having lost sight of the matter lately, through pressure of various business engagements, the writer is unable to say what measure of success this system has actually achieved.

In making a preliminary trial of the furnace, Mr. Brown stated that he was able to roast 50 tons a week, with a consumption of 1 cord of wood to 10 tons of ore roasted, and he expected to increase this by some alterations that were in contemplation to 70 tons weekly without any increase in fuel burnt, anticipating that 1 cord of wood would be sufficient to dead sweet roast 15 tons of ore.

(*To be continued.*)

NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE, APRIL 8TH, 1893.

Sir LOWTHIAN BELL, PAST-PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the last General Meeting and reported the proceedings of the Council at their meetings of March 25th, and of that day.

The following gentlemen were elected, having been previously nominated :—

MEMBERS—

Mr. M. W. BARKER, Engineer, P.O. Box 1,463, Johannesburg, Transvaal

Mr. HENRY B. BUNKELL, Engineer, P.O. Box 962, Johannesburg, Transvaal.

Mr. WILLIAM YOUNG CAMPBELL, Mining Engineer, etc., P.O. Box 357, Johannesburg, Transvaal.

Mr. WILLIAM CHARLTON, Mining Engineer, Linares, Provincia de Jaen, Spain.

Mr. SHERARD OSBORN COWPER-COLES, Manager, London Metallurgical Company, Limited, 80, Turnmill Street, London, E.C.

Mr. JAMES HUBERT DANGAR, Mining Agent, Camborne, Cornwall.

Mr. J. HUBERT DAVIES, Electrical Engineer, P.O. Box 455, Johannesburg, Transvaal.

Mr. ALFRED DUMAT, Mechanical and Railway Engineer, 1, Loop Street, Pietermaritzburg, Natal.

Mr. E. W. EVANS, Mining Engineer, P.O. Box 357, Johannesburg, Transvaal.

Mr. LOUIS FARAGHER, Mine Agent, Cape Copper Co., Ld., Ookiep, Namaqualand, South Africa.

Mr. ALBERT H. HALDER, Consulting Engineer, Pietersburg, Transvaal.

Mr. J. KERR HOULGATE, Mining Engineer, 69, Lowther Street, Whitehaven.

Mr. JOHN ARTHUR JONES, Mining Engineer and Analytical Chemist, Gijon, Asturias, Spain.

Mr. JOHN O'GORMAN LALOR, Mining Engineer and Metallurgist, P.O. Box 357, Johannesburg, Transvaal.

Mr. HENRY LAKIN LAWRENCE, Mining Engineer, Mines de Pierrefitte, Pierrefitte-Nestalas, Hautes Pyrénées, France.

Mr. RICHARD A. D. LIEBERT, Mechanical Engineer, P.O. Box 357, Johannesburg, Transvaal.

Mr. ROBERT H. LONGBOTHAM, Mining Engineer, Westgate, Wakefield.

Mr. DAVID ALEXANDER LOUIS, Metallurgical and Mining Engineer, 77, Shirland Gardens, London, W.

Mr. JOHN S. MACARTHUR, Metallurgist, 12, Knowe Terrace, Pollokshields, Glasgow.

Mr. John Hodgson Nicholson, Mining Engineer, Seaton Delaval Colliery, Northumberland.

Mr. Francis Daniel Peter, Mining Engineer, Greymouth, New Zealand.

Mr. John Gunsen Selkirk, Mining Engineer, Dalton-in-Furness.

Mr. A. Humboldt Sexton, Professor of Metallurgy, Metallurgical Department, Andersonian Buildings, 204, George Street, Glasgow.

Mr. Charles Liddell Simpson, Mechanical Engineer, Engine Works, Grosvenor Road, Pimlico, London.

Mr. Arthur James Stevens, Mechanical Engineer, Uskside Ironworks, Newport, Monmouthshire.

Mr. W. Henwood Trelease, Chief Agent of the Pestarena United G. M. Co., Ld., Pestarena, Vall'Anzasca, Novara, Italy.

Associate—

Mr. Joseph Thompson, Overman, 38, Church Street, Murton Colliery.

The following gentlemen were nominated for election :—

Members—

Mr. H. M. Becher, Mining Engineer, 9, D'Almeida Street, Singapore.

Mr. J. C. S. Beynon, Civil and Mining Engineer, P.O. Box 1,364, Johannesburg, Transvaal.

Mr. George Davey, Mining Engineer and Metallurgist, Ferro-carril Michoacan y Pacifico, Maravatio, Mexico.

Mr. Stanley H. Ford, Mine Agent, Stolzenfels, Great Namaqualand, South Africa.

Mr. Thomas Gilchrist, Mining Engineer, Manor House, Penshaw.

Mr. George Linday, Colliery Manager, Blackett Colliery, Haltwhistle, Northumberland.

Mr. William Ernest Lishman, M.A., Viewer, Bunker Hill, Fence Houses.

Mr. J. S. Quirk, Metallurgist, St. Helens Lead-Smelting Works, St. Helens, Lancashire.

Mr. F. G. Shaw, Mining Engineer, London.

Mr. Edmund Spargo, Mining Engineer, 3, Cable Street, Liverpool.

Mr. Charles Robert Western, Engineer, Broadway Chambers, Westminster, London, S.W.

Mr. Joseph Henry Woolcock, Civil and Mining Engineer, 49, Lowther Street, Whitehaven.

Associate Members—

Mr. G. A. Ferguson, Editor of the *Mining Journal*, 18, Finch Lane, London, E.C.

Mr. Henry M. James, General Manager, Colliery Office, Whitehaven.

Mr. Harry Page Woodward, Government Geologist, Perth, Western Australia.

Associates—

Mr. James Bainbridge, Under Manager, North Walbottle Colliery, Newcastle-upon-Tyne.

Mr. Robert Emerson, Engineer, Tudhoe Colliery, Spennymoor.

Mr. Alexander A. James, Under Manager, Croxdale, near Durham.

Mr. John W. Thompson, Colliery Surveyor, Backworth Colliery, Newcastle-upon-Tyne.

AWARDS FOR PAPERS.

The SECRETARY read the following lists of papers for which prizes had been awarded to the authors:—

YEAR 1890-91 (Conclusion).*	
NAME.	TITLE OF PAPER.
M. Walton Brown ...	Experiments with Explosives used in Mines.
M. Walton Brown ...	The Waddle Patent (1890) Fan.
W. Cochrane	On a Duplex Arrangement of Centrifugal Ventilating Machines.
J. Kenneth Guthrie ...	Pumping Appliances at Eltringham Colliery.
A. L. Steavenson	Notes on the Present Position of the Question of Transmission of Power.
YEAR 1891-92.	
P. Phillips Bedson and W. McConnell	Notes on the Gases enclosed in Coal and Coal-dust.
D. Selby Bigge	The Practical Transmission of Power by Means of Electricity and its Application to Mining Operations.
M. Walton Brown ...	The Rateau Ventilator.
L. W. Chicken	Experiments with Explosives.
W. Garnett	The Electric Transmission of Power and its Analogies in Hydraulics.
E. Halse	Notes on the Occurrence of Manganese Ore near Mulegé, Baja California, Mexico.
E. Halse	Notes on the Occurrence of Manganese Ore near the Arenigs, Merionethshire.
H. D. Hoskold	Notes upon the Mines in the Argentine Republic, South America.
J. D. Kendall	The Iron Ores of Spain.
J. M. Liddell	The Witwatersrandt Gold-field.
W. J. Orsman	Notes on the Products and Temperature of Detonation of some High Explosives.
W. J. Orsman	The Detonation of High Explosives by Percussion.
H. Palmer	Notes on an Electric Transmission Plant at East Howle Colliery.
J. L. Shaw...	The Hæmatite Ores of Cumberland.
T. H. M. Stratton... ...	An Enquiry into the Cause of the two Seaham Explosions, 1871 and 1880, and the Pochin Explosion, 1884.
R. T. Swallow	Description of a Method of Removing Deposits from the inside of Rising Main Pipes in Shafts.

* See also *Trans. Fed. Inst.*, vol. ii., page 472.

DISCUSSION UPON MR. JAMES LESLIE SHAW'S PAPER ON "THE HÆMATITE ORES OF CUMBERLAND."*

Mr. J. D. Kendall wrote that as Mr. Shaw had tacitly refused to abide by the decision of a third party in the matters of fact in difference between himself and the writer, there was, of course, an end to argument based on premises involving these disputed facts. There was still something to be noticed in Mr. Shaw's latest reply.† It was most difficult to make out with anything like certainty what Mr. Shaw wished members to understand from this contribution to the discussion. One thing, however, seemed fairly clear, *i.e.*, that he had completely abandoned the cavern explanation with which he set out. When endeavouring to account for the shale partings in the ore, he said‡ "the wearing action on the limestone, and the deposition of iron may have gone on simultaneously . . . . so that while the limestone was being softened and worn by acidulated water . . . . at the same time and from the same water iron was being deposited." Again, he said,§ "the two processes of wasting and deposition may have gone on simultaneously in all cases." Unless we are prepared to give up the accepted meaning of the word "cavern," it surely never can be said that this is a description of a cavernous deposition. It, to all intents and purposes, describes a replacement, but whether chemical or physical, it was impossible to make out, as Mr. Shaw had not yet thought it worth while to enlighten us as to the particular mode in which he conceived the ore to have been deposited, notwithstanding the writer's repeated requests that he would do so. Once again, will Mr. Shaw tell us in what chemical or physical form the iron was, that he supposed to have been deposited from the acidulated water, and how it was, in the first instance, separated from the enormous amount of siliceous and argillaceous matter with which he assumed it to have once been associated? As an excuse for not referring in his original paper to interbedded shales, Mr. Shaw said ‖ that he "had never seen the necessity of bringing in shale-beds to support his opinion on the formation of hæmatite, and he did not therefore think it necessary to ask Mr. Kendall where the shale-beds . . . . were to be seen." Surely this statement, if meant, showed that Mr. Shaw lacked true scientific method. Writers in search of truth—and not merely anxious to bolster up a preconceived opinion—put forward all the facts, and not simply those that accord with their hypotheses. If Mr. Shaw knew anything of the

* *Trans. Fed. Inst.*, vol. iii., page 580. † *Ibid.*, vol. iv., page 147.
‡ *Ibid.*, page 147. § *Ibid.*, page 149. ‖ *Ibid.*, page 148.

literature of hæmatite he must have known that shale-beds were looked upon (by some writers at any rate) as an unanswerable argument against the cavernous explanation of hæmatite deposition, and, seeing that he adopted the cavernous explanation in his paper, he was morally and scientifically bound to notice the shale-beds and reconcile their existence with his theories. Why Mr. Shaw should refer to the Mammoth Cave of Kentucky was inconceivable, seeing that in his latest reply he avoided the necessity for caverns; but it might increase his confidence in his new position to be told that the span of the largest chamber in the Kentucky cave was not one-third of that of the Gillfoot Park deposit adjoining the cemetery. Notwithstanding that fact, its floor was covered with enormous blocks, which had fallen from the roof; whereas, there was not a particle of evidence that any stone had fallen from the roof of the Gillfoot Park deposit. Mr. Shaw, it appears, now* assumed that the writer had no previous acquaintance with the Gillfoot Park deposit, because the section accompanying his (the writer's) first remarks upon Mr. Shaw's paper was not drawn to scale. If Mr. Shaw's other conclusions were based on premises so slender, it was not surprising that they were lacking in accuracy. When the writer sketched the section referred to, it would have been quite possible for him to have drawn that section to scale. It was, however, unnecessary, as the section was simply intended to illustrate the parallelism of the bedding-planes in the limestone and shale to the ore-bed. The expression "growing together" was still objected to by Mr. Shaw, but he did not suggest a better one, nor show how its employment might lead to misconceptions. It was an expression in common use among miners, and, in the writer's opinion, one that expressed the fact tersely. Mr. Shaw's reference to the formation of flints in explanation of his phrase "affinity of particles" reminded one of attempts that were sometimes made, in another region of thought, to explain phenomena by assumptions—the assumption being as difficult to understand as the phenomena themselves. If "affinity of particles" meant the same as the "mechanical attraction of particles," what did the latter mean, and how did it operate in the formation of iron ore?

The CHAIRMAN then declared the discussion closed.

Mr. A. L. STEAVENSON read the following paper by Mr. Edward S. Wight, on "Queensland Coal-mining; and the Method Adopted to Overcome an Underground Fire":—

* *Trans. Fed. Inst.*, vol. iv., page 148.

QUEENSLAND COAL-MINING; AND THE METHOD ADOPTED TO OVERCOME AN UNDERGROUND FIRE.

By EDWARD S. WIGHT.

This paper is written with the object of giving some little idea of coal-mining in Queensland, and the trouble experienced from underground fires. The writer trusts it may prove instructive to some of the members, and that a more graphic account may be presented at some future time.

The coal-seams lie in the Permo-Carboniferous formation, and are both numerous and thick, covering a large area around Bundamba and Ipswich. Coal has also been found in various other parts of the colony, including Jimbour, Townsville, Cooktown, and Nosa, but this paper will be confined to the Ipswich coal-field, as the writer has not visited any of the others: it excludes also the Burrum coal-field, near Maryborough, where a large quantity of coal is raised, and where the systems of working are much the same.

The most successful mines are owned by private individuals, who either own the land or have leased the coal at a royalty of about 6d. per ton. The owners or lessees manage their own mines and perform part of the manual labour besides, as the price obtained for the coal will not pay the high salaries for manager, secretary, shipping agent, office rents, and directors' fees necessitated by properly formed companies; to make a profit, all must be workers.

The principal collieries on the south side of the Bremer river, which divides Ipswich, are the New Chum, Lewis Thomas, Stafford Bros., and West Moreton. The shaft at West Moreton colliery, the deepest in the colony, is 640 feet deep: it is of oblong shape, 15 feet by 6½ feet, divided by a wooden brattice, 8 feet being used as a winding and a downcast shaft, the remainder being used as an upcast shaft. The winding engine has two direct-acting cylinders 22 inches in diameter, with two drums 11 feet in diameter, one of which is fitted with clutch gear. The pulleys, 11 feet in diameter, are set on a substantial and well-designed hardwood framing. The cages are two-decked, and carry two 7 cwt. tubs on each deck. This is the best winding-plant in the colony. The seam worked is 19 feet in thickness, dipping westward at the rate of 1 in 3; only 7 feet of this seam

is at present worked, containing one stone band 8 inches thick; 5 feet, however, above this is workable coal if only proper means of cleaning it were erected, such as the belt screens lately described in the Transactions of this Institute. The coal contains about 7 per cent. of ash, and burns freely and well for both steam and household purposes. The coal is seldom screened, the system adopted being that of riddling underground, and leaving the slack in the mine, as there is no market for the whole of it. The price paid to the miner is 3s. 4d. per ton, which prevails all over the district; for this wage the miner gets and riddles the coal, sets his own timber, lays the road and wheels the coals, sometimes a distance of 300 yards. The miners thereby earn 9s. to 11s. per day of 9 hours, while the rate paid for taking a miner off the coal to datal work is 8s. per day; other wages are about 7s. per day; mechanics, 8s. to 9s. per day; and boys, 3s. to 6s., according to size.

The prevailing method of working is bord-and-pillar, leaving small pillars to crush, with no intention of taking them out. But some of the mines, with the seam at a less inclination, leave larger pillars which may be worked when coming back. In place of brattice, the miners form an intake and return airway by gob, composed of the slack left in the mine and kept up at the roadside by slabbing and props.

The slack is liable to spontaneous combustion, and the gobs should either be levelled over after the bords are done, or each district securely sealed up; the latter plan is probably the better, as it enables air-currents to be shortened.

An attempt was made by Mr. Lewis Thomas to utilize the slack in the manufacture of fuel briquettes; he erected a plant and made a few hundred tons of fuel, but owing to the high cost of pitch the system was abandoned.

No attempt has been made to produce different sizes of coal; the two varieties being simply riddled coal (a $\frac{3}{8}$ inch riddle being generally used) and slack.

The largest output from any one mine is 500 tons per day, but this has not been obtained from one shaft. There are numerous small mines opened by working men, who are afforded a splendid opportunity of commencing for themselves in this way by the Government providing all rolling stock; and in fact nearly all the mines have been thus opened out by working men.

The working of the mines is controlled by an Act of the Queensland Parliament, which came into force at the beginning of 1890, the principal clauses being:—Formation of two means of outlet; the use of

detaching hooks; the formation of special rules; moreover, enginemen must now be certificated, and are not allowed to work more than eight hours (in this respect it is a step in advance of the English Mines Act, and a step too far when there is no compulsion with respect to the management).

The coal on the south side of the Bremer river is non-coking, but on the north side another series of seams occur from which a very good coke is made and finds a ready market. Like those on the south side, the mines have all been opened without capital, and their development depends entirely on profit. The writer believes that a proper system of crushing, sizing, and washing of the coal would result in its yielding a coke which would drive the English article out of Australia. Very few of the north-side mines are connected with the railway, and depend entirely on river transit.

The produce is taken down the river in punts carrying 100 tons to 150 tons. The river transit is, moreover, much cheaper than rail transit, the cost of the former being about 2s. 6d. per ton to put the coal on ship-board, whereas the cost to the south side mines is as follows:—Average railway freight, 1s. 6d.; wharfage, *i.e.*, putting the coal from truck into punts at Brisbane, 7d. per ton; puntage and putting the coal on ship-board, 2s. per ton; total, 4s. 1d. per ton.

The selling price ranges from 10s. to 11s. per ton for shipping. The inland trade is the most profitable when there is no puntage to pay. The reason of the puntage is that there is no export trade, the market being the bunkering of the coasting steamers, which usually stay only a few hours and take in coal while discharging cargo.

The mines are worked exclusively with naked lights, carburetted hydrogen being unknown. The only gas met with is carbonic acid and that in small quantities, which a good air-current easily removes. Furnace ventilation is universally adopted. The natural heat of the mines is great (the writer had not, it is true, any means of properly testing it), and the mine, 3 or 4 yards from the main air-current, however good this may be, is unbearably hot.

Mining accidents are not numerous. At West Moreton colliery, already referred to as the deepest in the colony, which has been working for three years, there has not yet been a fatal nor even a serious accident.

Underground Fire at West Moreton Colliery.

Fire had broken out underground before the writer took charge of this colliery, caused by the spontaneous combustion of the small coal; it was awkwardly situated, as it closed the best district, reducing the output

by 100 tons per day, and also rendered useless the second means of exit obtained by holing to the adjoining colliery. The fire was confined to one roadway, 40 feet above the level of the shaft, and the indiscreet way in which the workings had been driven, and the absence of a correct plan, rendered the matter of dealing with it one of difficulty. The means adopted were to take water from the surface, obtained from a cistern into which the mine water drawn by the cages was discharged; 2 inches pipes were taken underground to the scene of the fire, which could only be approached from the return side; a brattice was run up to convey air enough to enable the men to work, and the water was discharged through a nozzle on to the burning mass, which, as it cooled sufficiently, was filled away in skips and sent to the surface. Good progress was made in this way, and no doubt the fire would have been eventually subdued had not the heat and smoke, with the little fresh air which had to be supplied to the men at work, ignited the coal in another place in the return airway.

At this stage it was decided to seal up the district; in the absence of a plan of the workings, and, being a stranger in the pit, the writer was entirely dependent on the overman, who was the walking plan, to decide where the stoppings would have to be placed to exclude all air from the district. After investigation it was found that seven stoppings would be required, four in narrow places and three in wide bords; the same day the air was shut off by temporary stoppings of small coal in the wide bords and plastered wood stoppings in the narrow places. The presence of large volumes of black-damp rendered the work difficult, but all was made ready for the permanent stoppings of 9 inches brickwork and the erection was hastened as much as possible. No great difficulty was experienced until the building of the last stopping, which was 15 yards long, all the material for which had to be hauled 100 yards by a windlass up a gradient of 1 in 3. At this point large quantities of black-damp began to percolate through the temporary stopping, and all available air had to be conducted to the face of the work to enable the men to work even for short periods. It was, however, successfully completed, and after the mortar had dried and the brickwork was re-pointed, not a particle of black-damp could be detected.

It was the writer's intention, after a lapse of twelve months, to endeavour to drain off the black-damp and re-enter the district, but the financial condition of the company prevented him from being there to carry out the operation. The writer knows that the stoppings held good and that no further trouble was experienced from the underground fire.

Sir Lowthian Bell, having to leave at this stage, Mr. W. Cochrane took the chair, and invited discussion on the paper.

Mr. T. E. Forster said he was perhaps the only person in the room who had been down a colliery in Queensland, and his experience did not include the Ipswich coal-field described in the paper, the one he had visited being that of the Burrum river. He would reserve any remarks he might have to offer until he had an opportunity of reading the paper carefully.

The discussion was then adjourned, and a vote of thanks was accorded to the writer of the paper.

Mr. R. A. S. Redmayne read the following paper on "The Geology and Coal-deposits of Natal":—

THE GEOLOGY AND COAL-DEPOSITS OF NATAL.

By R. A. S. REDMAYNE.

Introduction.

After a residence of about nineteen months in this interesting colony, during which period the author prospected several properties and traversed the greater part of Natal from south to north and east to west, he ventures to record the results arrived at and opinions formed regarding its geology during his stay. He also desires to express his appreciation of the courtesy of his late directors in permitting him to read this paper, and to acknowledge his indebtedness to his friends, Dr. Sutherland, the ex-Surveyor-General of Natal; Captain G. Nicholls, R.E., Her Majesty's Commissioner of Mines in Natal; and to Mr. Dumat of Pietermaritzburg, for much information kindly afforded.

Physical Features.

From the Drakensberg range the land descends by steep terraces and gentle slopes to the sea-level. Natal, in fact, may be regarded as comprising three distinct plateaus, and as having corresponding variations of climate and production. These are: (1) the low-lying riviera extending inland for 15 to 20 miles, which is semi-tropical, producing tea, sugar, a little coffee, bananas, etc.; (2) the midlands, which are cold in winter, (Pietermaritzburg being regarded as the representative town): here oranges and vines flourish; and (3) the uplands, forming the chief grazing-grounds for sheep, cattle, and horses. This country is nearly destitute of trees, and is known as the "winter veldt," for the Boers bring their sheep and goats down in winter from the high veldt of the Free State to graze them in this, the warmer country. Ladysmith and Newcastle are the chief towns of the upland district.

The colony is divided into nine counties (Fig. 1, Plate XXII.), four of which, Victoria, Durban, Alexandra, and Alfred, are on the coast. The country is full of valleys and hills, and, although there are numerous watercourses, many of which, it is true, contain very little water in the dry or winter season, there is no navigable river.

The peculiar flat-topped or table-like mountain is a very characteristic feature of the Drakensberg range, and is due to the doleritic outflows which cap the mountains and preserve them from denudation. Many of the perpendicular precipices, or "krantzes" are of the same formation.

Geological Sketch.

So far as the writer is aware, no general geological map of Natal has been constructed, nor has any geological memoir been compiled—if we except that brief, though admirable, little pamphlet read by Dr. P. C. Sutherland, the ex-Surveyor-General of the colony, before the Natural History Association of Natal in June, 1868.

Alluvial Deposits.

Commencing with the most recent formation, viz., the alluvial, there is in the promontory known as the Bluff, at Durban, on which the lighthouse stands, a raised sea-beach, which is 300 feet above sea-level. It consists entirely of sand and broken shells. The writer has been informed that many parts of the coast are of similar character. There can be no doubt that the area next the coast (shown in the section, Fig. 2, Plate XXIII., kindly given to the writer by Dr. Sutherland) has been recovered from the Indian Ocean, the waves of which at one time beat against the Berea Ridge, on which are now situated the charming suburban residences of the Durban merchants.

The solvent power of water containing carbonic acid gas is well demonstrated by the accumulation of nodular limestone which segregates in the soils in many parts of the colony. In the Tugela Valley (as Dr. Sutherland informed the writer) there are masses of this species of limestone several hundred tons in weight. It is the excess of the carbonic acid gas that dissolved the lime out of the original rock, and upon its escape the lime was deposited as a carbonate. Bog iron-ore, or limonite, so very common all over the colony, has also been deposited in like manner. The limestone and the iron-ore are, however, too impure to be of commercial value.

Tertiary System.

No portion of the Tertiary system is represented in Natal. It may be that the fossil wood so abundant along the Drakensberg range is of this age, but that is an unsolved problem.

Cretaceous System.

The first formation of any importance found in descending order is the Chalk, which is seen on the sea-coast at the mouth of the Impengati river

in Alfred county, and at the mouth of the Umzumbe river, about two miles south of the Umtamvuna ; some of the ammonites found in this formation are of considerable size. Large collections of fossils are to be seen in the Durban and Pietermaritzburg museums. Numerous magnificent forest-trees, some of which are 70 feet in length and 2 feet in diameter, are also found ; their cavities are filled with flint, a characteristic feature of Cretaceous deposits throughout the world. These Chalk deposits consist of greenish sands and clays with rolled pebbles, derived from the older rocks, the cementing-matter being calcareous.

The greatest elevation of the Chalk above the sea does not exceed 100 feet. In no place has the formation been traversed or interrupted by intrusive dykes or outflows, a fact worthy of note in a country so scarred with igneous intrusions.

Oolitic System.

The Oolitic system is absent from this part of the African continent, for between the Umtamvuna Chalk and the coal-formation no intermediate features are found.

Triassic or Coal-measure System.

The Triassic system is the carboniferous or coal-measure formation of Natal, and probably of South Africa generally. Commencing at the surface in the interior, there is a gritty sandstone lying in nearly horizontal layers, and alternating with various coloured shales of differing thicknesses. The maximum thickness of the series is probably not more than 4,000 feet. Dr. Sutherland considers that the rocks found on the top of the Town Hill at Pietermaritzburg form the base of this formation (and judging from personal observation the writer agrees in that view) ; thence the series rises to nearly the summit of the Little Drakensberg, which is over 7,000 feet high.

On the surface of this formation is found the fossil coniferous wood, which appears to be the same as that found in the Cretaceous deposits of the colony. The area over which this formation extends is enormous : from the Town Hill, Pietermaritzburg, to the Drakensberg range, and thence to the Orange Free State and the Transvaal.. It forms also a narrow strip in Victoria county from the Umgeni to Compensation.

One of the most remarkable features of this formation is the number of igneous dykes and outflows which intersect and cut up the series. A very good example of this is found in the Zwartkop mountains ; it may be traced (through a series of outbursts) into the Great Drakensberg, where it forms the weatherworn peaks, which rise to a height of nearly

10,000 feet above the sea-level. These dykes vary from a few inches to many feet in thickness, and may be many times repeated in a short distance. At the Howick falls (Umgeni river) near Pietermaritzburg, three distinct beds of greenstone are to be seen, separated by intervening beds of sandstone and shale.

The localities in which coal has been found are shown on the map of the colony (Fig. 1, Plate XXIII.); workable seams are, however, only found and worked within the shaded area in Klip river county. At the present time prospecting and borings are being made in the strip of coal-measure formation found in Victoria county, but from the facts and information so far to hand it does not appear that anything like a workable seam of coal has been or is likely to be discovered. The writer is, however, of the opinion that workable coal may possibly be found near the Drakensberg mountains, due east of Pietermaritzburg (Appendix A).

Mr. T. Bain and Prof. Seeley state in their report on the Geology of Cape Colony:—"At Cyphergat and Fairview, in Molteno, we obtained evidence that the fossil ferns and other plants which are found in shale immediately above the coal, are identical with those which occur in the same position in the Indwe coal-field," and that they also found remains of trees standing erect as they grew about the coal of Cyphergat, "thus demonstrating that the coal occurs in the position in which it grew."

After careful perusal of Mr. Galloway's report of 1890 to the Cape Government, and Prof. Green's paper of 1888, one cannot but come to the conclusion that the coal is of subaqueous origin. From the accounts given of Cape Colony coal-fields one cannot but be convinced that they are also contemporaneous with the Natal formations. The reasons advanced to prove the Natal coal-fields of subaqueous origin, are:—

(*a*) Fossil-remains are exceedingly scarce either just above, below, or in the coal. At Boston mills there is a 9 inches seam of coal resting on a bed of shale, in which roots of coal-plants are constantly found; and the roof, also shale, contains numerous leaves of coal-plants; while at a few other places, viz., Estcourt, Weston, on the Mooi river, and Lennoxton plant-remains are found, but as a general rule they may be considered as extremely rare. The finding of fossils in a few isolated cases does not militate against the subaqueous theory.

(*b*) The extraordinary variations in the thicknesses of the seams of coal in a very short distance: in one case thinning out to 1 foot and in a few yards increasing to 3 feet, and again in the same distance thinning out again. The wavy and false-bedded nature of the strata immediately above and below the coals, and the great variation in the thickness and

character of the bands separating the coal, as well as the ripple-markings, so common in the sandstone or shale just above and below the seam, are further proofs of the origin of the deposits.

(*c*) The total absence of any underclays. The floor of the seams being usually sandstone, and sometimes a shale-bed.

(*d*) The instance mentioned by Dr. Sutherland, where, in the Newcastle district, there was found embedded in the coal a fragment of granite, which weighed about $1\frac{1}{2}$ lbs., goes further perhaps than anything else, to prove these coals to be drift material. Dr. Sutherland himself, however, does not bind himself to any theory.

(*e*) The specific gravity of the coal is high, showing the presence of earthy impurities, which is borne out by chemical analysis, and on very close examination thin layers of earthy matter are observable in some of the coals.

The most characteristic fossil of the few found in the coal-measures, is *Glossopteris*, which is eminently a Triassic fossil. No remains of *Stigmaria* or *Sigillaria* have as yet been discovered, and the writer has only seen one specimen of *Lepidodendron* (found near Newcastle, S.A.).

Pietermaritzburg Shales.

Below the sandstone and Carboniferous shale formation, referred to above, come the Pietermaritzburg Shales, which may be taken as contemporaneous with the Kimberley or Olive Shales (described by Prof. Green in his paper). This shale-formation, which is found all over Pietermaritzburg, can be traced without a break nearly up to the summit of the Town Hill. No indications of plant or other reliable fossil-remains are found in it. These beds may be regarded as representing the Middle Trias.

*Boulder-clay Formation.**

The Pietermaritzburg Shales pass downward into a very curious formation of breccia, which has, so far, proved the *bête noir* of South African geologists. It is composed of a greyish-blue argillaceous matrix, containing numerous fragments of igneous rock; these vary greatly in size, some being very large, weighing upwards of 5 tons, uniformly dispersed throughout. Dr. Sutherland says "It passes on one side only into the Pietermaritzburg Shale, and on the other it abuts upon the next succeeding formation underneath, upon which it appears to rest." The included fragments of rock have the appearance of being smooth and polished by attrition.

* The Boulder-clay formation is a very misleading term for a Palæozoic group, and the author regrets that he is compelled to make use of it.

Dr. Sutherland calls it the Boulder-clay formation. Mr. Dunn describes a similar deposit in Cape Colony, which he names the Dwyka Conglomerate; Mr. Bain, the Claystone-porphyry; and Mr. Wyley, the Trap Conglomerate, meaning thereby that it is of volcanic origin, which is by no means certain.

Prof. Green admirably describes the rock as it occurs at Grahamstown (Cape Colony), and his description tallies exactly with the character of the Natal deposit. He speaks of it as "a dark-grey, closely-grained, hard matrix, thickly set with many small angular fragments and rounded grains of quartz, the larger included blocks not very numerous, the majority angular and subangular, but some well-rounded pebbles . . . . a whitish granite, rich in quartz, was by far the commonest rock among the fragments, but quartzites and other rocks occur. In many places no bedding is discernible, but where the rock has been exposed it has a bedded look . . . . The matrix of the rock" he adds "has all the look of mashed-up granite mixed with fine mud." Although Prof. Green says "the size and angularity of the boulders suggest the action of ice," he leans to the opinion that it is a lacustrine deposit. Dr. Sutherland, in a letter to the writer, says "There can be no doubt that the formation in question had its fringe exposed to the action of water. At the point where it passes into the shale it is in many parts ripple-marked, which is clear proof of its exposure to the action of water subject to oscillation of the surface with regard to the beach, as well as back-and-forward motion. These ripples are best seen at Umzinto, where good puddingstone is won from the formation at the line of its passage into the shale."

The area over which this formation extends is very great. It is found throughout the northern part of Victoria county; it flanks the sandstone hills on their seaward side from the Tugela river to the Umgeni river, and occupies both sides of the Umgeni river at Queen's Bridge; it passes south through Cato Manor and the Berea, and crops out at the sea-level at the Umbilo, near Congella. It is further to be found extending as far as the St. John's river. On the other hand, it is found cropping out extensively in the vicinity of Pietermaritzburg, between Greytown and the Biggarsberg mountains, and many other places.

The equivalent of the Old Red Sandstone is wanting in Natal.*

Silurian System.

Below the breccia we pass into a sandstone of Silurian age. These Silurian rocks occupy the flanks and hollows of the primitive rocks on

* The author is inclined to think that the Dwyka Conglomerate represents a part of the true Carboniferous system.

which they rest horizontally and much faulted. In many parts they contain numerous rolled, polished fragments of quartz derived from these older rocks. The thickness varies up to 500 feet in Natal, though it is found much thicker in other parts of South Africa. Table Mountain (near Pietermaritzburg), Inanda, and Izwate are instances of it.

Pre-Silurian Rocks.

The primitive rocks, consisting of granites, mica-schists, clay-slates, and marbles, are seen in the gorges of the mountain-valleys, and at Bishopstowe, close to Umsumdusi river. The series extends northward through the Inanda location into Zululand, following the deep valley of the Tugela river. At the Lower Umzimkulu river, about 6 miles from the sea is found a mass of metamorphosed limestone, forming a marble of great purity, but too highly crystallized for statuary purposes. This deposit is of very considerable extent, having been proved to exist over 35 square miles, and it is not less than 1,200 feet thick.

Summary.

The following classification of the Natal rocks may, the writer thinks, be adopted, and is shown in correlation with Prof. Green's division of the Cape Colony rocks :—

Natal.	Cape Colony.	
Alluvial	Alluvial.	
Cretaceous	Cretaceous.	
Igneous rocks, dykes, and outflows, capping the Drakensberg range ...	Volcanic beds.	
Coal-measures (Triassic)	Cave sandstone.	Stormberg beds.
	Red beds.	
	Molteno beds.	
	Karoo beds.	
Pietermaritzburg Shales	Kimberley Shales (great unconformity).	
	Ecca beds.	
Boulder-clay	Dwyka Conglomerate (great unconformity).	
Silurian	Quartzite.	
	Bokkveldt beds.	
	Table Mountain sandstone.	
Pre-Silurian (gneiss, granite, marble)	Slates and intrusive granite (Malmesburg beds).	

The Northern Coal-fields of Natal.

It has been shown that coal undoubtedly exists in many parts of Natal, but readily workable seams occur only in the Klip river district.

Mr. F. W. North, commissioned by the Government of Natal to prospect the coal-fields of the colony, reported thereon in September, 1881. His report undoubtedly led to the erroneous notion of the existence in the Klip river district of an extensive workable coal-field abounding in outcropping coal-seams "invariably situated in favourable positions for economical working."

In reality, the coal-deposits in Natal are spread about in beds of coal of variable quality of no great extent individually, and often reduced by erosion. In the Klip river district, for instance, coal-seams outcrop over a country of about 1,190 square miles, but this very considerable area is no gauge of the workable coal-beds. The assertion of their being invariably situated in favourable positions for economical working is disposed of, when the aspect of that broken country is considered.

Since the issue of Mr. North's report, the Natal Government Railway has been completed to the frontier of the Transvaal. (Fig. 1, Plate XXIII.) Laterally to the line, but only within the limits of a comparatively narrow belt where branch-lines can be constructed at reasonable cost, has the coal country been opened out; otherwise it remains as inaccessible as ever. Any other railway through it, if ever constructed, will be attended with similar limited advantages.

The belt of coal-bearing lands along the railway line from Elands Laagte to Newcastle (Fig. 1, Plate XXXIII.) can be divided into three parts:—(*a*) south of the Biggarsberg range, (*b*) the Dundee plateau, and (*c*) the Newcastle district.

Biggarsberg Coal-field.

In that part south of the Biggarsberg range and nearest the seaport, the apparently promising coal farms were hurriedly taken up in expectation of immense financial profits, were partially developed, and subsequently nearly all abandoned, owing to the inferior quality of the coal, the thinness of the seams, or the impossibility of connecting them to the main line of railway at a cost in proportion to the importance of the mine; and unless some improbable circumstance intervenes they are not likely to be re-opened.

A brief description, however, of these mines may not be out of place.

(1) *The Elands Laagte Mine.*—This was one of the first mines which produced coal in the colony. During the year 1889, 19,094 tons were raised, and 7 Europeans, 2 Indians, and 100 natives employed. But in 1890–91 the output fell to 7,334 tons, and the mine has now ceased working. The coal supplied from it had no very good reputation in

the colony, owing to its high percentage of sulphur, and its being put on the market in a dirty condition.

(2) *The Natal Colliery.*—The output from this colliery for the year 1889 was 2,206 tons, the labour employed consisting of 2 Europeans and 18 natives; the output in 1890–91 was only 373 tons. It is now closed.

The section of the seam was as follows:—

	Ft.	Ins.
Sandstone-roof.		
Coal	2	2
Shale-band	0	9
Coal	1	4
Sandstone-floor.		

The Shotts colliery, a small mine, was opened on a portion of the Natal Colliery Company's property, the coal produced from it (in 1891) being 142 tons.

(3) *The Ramsay or Inkunzi Colliery.*—The writer spent some time at this mine, over which he was conducted by the then manager, Mr. Tylor, who was at one time a member of this Institute, and who served a portion of his apprenticeship in Northumberland. The writer can safely say that it was the only properly laid-out and best managed coal-mine in Natal.

The seams of this neighbourhood are mentioned by Mr. North on page 29 of his report, and shown in his sections 54A and 55. The writer obtained the following section from Mr. Tylor:—

	Ft.	Ins.	Ft.	Ins.
Sandstone, increasing in thickness as the hill rises.				
(1) Not worked—				
Coal	3	0		
Shale	1	6		
Coal	1	0		
Sandstone, false-bedded, solid above shale	20	0		
Shale	2	6 to	4	6
(2) Worked—				
Coal	1	8		
Parting	——			
Band (average 2 inches)	0	1 to	0	6
Coal	0	8		
(3) Not worked—				
Sandstone	12	0 to	18	0
Coal	1	0		
Splinty coal	0	6		
Sandstone.				

The mine is worked by drifts or adits driven in the coal which crops out near the base of the hill. There are dykes in the vicinity of the mine, but none of these have been passed through, for the workings have not proceeded far enough as yet.

The coal being worked was more like good North-country (Durham) household coal than any the writer has seen in the colony ; it is also more bituminous than most of the Newcastle (Natal) coals. When the writer visited the mine it was being worked on the longwall system, by far the most suitable method of working a seam of this nature ; the shale-roof being ripped down in the gateways to make tub and travelling-height. The squeeze or subsidence was very even, and closely followed the advancing workings. The surface appeared to be in no way damaged.

The seam was kirved just above the band, the top-coal being wedged down and the bottom-coal wedged up. All hewing and transport was performed by 24 Kafirs, the only white man being the manager. The hewers worked from sunrise to sunset, with intervals for meals amounting to two hours in all, and were paid at the rate of £2 5s. per 30 working days, with rations (in addition) of mealie meal, and meat once a week. The night "boys," who rip down the shale for height, worked 14 hours with no allowance for meals, and received 5s. per month (30 days) more than the hewers. The cost of working and conveying the coal to the mouth of the adit was 7s. per ton, on an output of about 15 tons per diem.

The output for 1889 was 2,500 tons, and from July, 1890, to 30th June, 1891, 4,704 tons were worked. The coal was conveyed to the Natal Government Railway, 3½ miles distant, in Scotch carts drawn by oxen in summer and by mules in winter. The cost of transport being 3s. per ton.

The mining rights were leased from one of the Amakowlwar Kafirs (Christians), who own and farm the land in this district. This mine has recently been closed.

(4) *The Acol Colliery.*—The seam worked, which is of the following section, corresponds to the upper or No. 1 Seam of the Ramsay mine:—

	Ft.	Ins.
Sandstone-roof.		
Coal	1	10
Shale	0	8
Coal	1	4
Sandstone-floor.		

During 1889, 600 tons were worked, and the output increased during 1890-91 to 4,235 tons ; but very little, if anything, is being done at the mine at the present time.

(5) *The Wallsend Mine.*—The Natal Government Railway passes through this property. The section of the seam worked was:—

	Ft.	Ins.
Sandstone-roof.		
Shale	1	9
Coal	1	10
Shale	2	5
Sandstone-floor.		

The output for 1889 was 184 tons, and in 1890-91, 2,000 tons; the labour employed being 7 Europeans and 70 natives. The coal on this property is extremely good, and the best results were obtained from it when it was used on the Government Railway; but owing to the thinness of the seam, the cost of working it was too high to allow of the carrying on of the mine at a profit, and it ceased working in 1891.* Recently, however, the Uithoek, a small mine, has been commenced in the northern part of the farm.

The Dundee Plateau.

North of the Biggarsberg range is the Dundee plateau. Here the successful Dundee colliery (11,000 acres, capital £110,000) has absorbed a great part of the workable coal-beds of the neighbourhood, and the St. George's Company contemplate starting, some miles to the north. That company having, the writer understands, completed the necessary survey and proved their property.

The Dundee Colliery.—The writer several times visited this, the premier mine of the colony, the workings of which he surveyed for the Natal Government Railway authorities. The following is an average section of the seam worked:—

	Ft.	Ins.
Sandstone-roof (firm).		
Coal	1	2
Shale (taken down to make height in main roads)	0	3
Coal (worked)	4	8
Shale	0	4½
Coal	1	10
Sandstone-floor.		

There are three shafts, coal being drawn from two of them.

The seam lies at a depth of 60 and 70 feet in the two working shafts, respectively, the superincumbent stratum being sandstone, which reaches to the surface. The shafts are fitted up with rail-guides and single-decked cages, capable of holding two 10 cwts. tubs. The drawing is performed by two small Robey winding-engines. The mine produces a quantity of water, which is raised to the surface by a small double-acting Tangye force-pump.

The workings are laid out and conducted on the old-fashioned bord-and-pillar system. The pillars or stoops are very small (5 by 3 yards). The galleries average about 4 yards in width. Little or no use seems to have been found for the slack or small coal which lies in large heaps about the surface.

* The writer is informed that the mine has recommenced working.

One of the main roads is driven through a large dolerite dyke, the coal in its immediate vicinity being much burnt, but very little disturbed. The writer observed this latter peculiarity throughout the coal-field.

The mine is connected to the Natal Government trunk-line by a single line 7 miles long, which, although laid over a level country, with few impediments in the shape of spruits or dongas, cost about £58,000. The Government are taking the line over from the company at the same price, the payment being spread over five years.

When the writer visited the mine in August, 1891, there were employed 24 Europeans, 140 natives, and 35 Indians, and the output amounted to about 3,500 tons per month. The company now employs about thrice as many natives, and raises about 10,000 tons per month.

On the commencement of removal of the pillars in 1892, a large subsidence of the surface took place.

Cartridges of gunpowder mixed with sawdust are used for bringing down the coal, they are ignited by means of fuse. The kirving is made in the coal. Candles are used by the miners (2 tons per month), but the electric light is in course of being installed at the shaft bottom and in all the main roads. A ventilating-fan is also being established at No. 3 shaft.

The cost of working at this colliery in the year 1890, according to Mr. Hallimond (the then manager) was under 5s. per ton.

The wages for Europeans varied, and depended on the character of their work; so far as the writer can remember them, they were as follows:—Overmen, £16 to £20 per month with free house and coals. Other white labour, from £10 to £14 per month. Kafirs, £1 10s. to £1 15s. per 30 working-days with mealie-meal rations and meat every Saturday. Corrugated-iron barracks are provided for the natives to sleep in. The Indians, who are employed about the surface, are paid a somewhat lower wage than the Kafirs, and receive, by way of rations, rice and dohl (dried fish). Masons' wages are very high, and most of their work is done by contract.

Newcastle District.

In the Newcastle district there are, within easy distances from the main line of railway, working and workable coal-deposits of very considerable value, and it is the writer's belief that the district will eventually prove to be the chief source of coal supply for South Africa; at any rate until the Zululand coal-deposits are worked.

Fig. 3, Plate XXIV., shows the relative positions of these properties and their proximity to the Natal Government Railway.

The Southern Cross, Newcastle colliery, and Rooi Pont estates have been proved, partially developed, and are working. The Ballengeich farm has been proved by borings, discovery of outcrops, and a shaft has been sunk. The Braakfontein, Tiger Kloof, and Leicester farms, and two blocks of Government land (42 and 111) all contain sufficient indications of coal to justify the expense of proving them.

The other coal-bearing farms of the district, including some of the Government lands, have not been considered, either by reason of the bad quality of the coal, or the impossibility of connecting them with the main line of railway at a cost in proportion to their present importance.

The writer believes that the amalgamation of these properties under one colliery and land company, with adequate working capital, would, for a long time, have rendered the formation of any new parallel operation at least improbable; and, perforce, would have made them the sole competitors of the Dundee collieries for supplying the local and export trade.

(1) *The Southern Cross Colliery.*—This company possesses four blocks, I, J, K, M, of the Newcastle town lands.

Fig. 4, Plate XXIV., shows the configuration and area of the property, the position of the boreholes and of the shaft, by which the seam is now worked, and the position of the outcrop on the left bank of the Ingagane river. The sections of the several boreholes are noted in Appendix B.

The following is a detailed section of the upper seam now being worked, chiefly for household purposes :—

	Ft.	Ins.
Black-shale-roof	1	0
Splint	0	7
Coal	2	3
Sandstone-band	0	1
Shale-band	0	3½
Coal	0	1
Splint	0	6
Coal	0	7
Splint	0	3
Coal	0	3
Splint	0	2
Coal	0	6
Total thickness	5	6½

According to the borings, the coal extends in workable thickness over at least 273 acres in the upper seam at a depth of 54 feet, and over an equal area in the lower seam at a depth of 81 feet. The average thick-

ness of the seams can be reckoned at 3 feet 5 inches, and after a deduction of 5 per cent. for loss underground, there will be 1,200,000 tons of available coal in each seam, of which three-fourths may be reckoned as round coal and one-fourth as small coal and dust.

Samples of the coal have been analysed by the chemist of the South African Trust and Finance Co., Ld., of Johannesburg, showing:—

Moisture	1·91 to 2·27
Volatile products	34·49 „ 36·73
Fixed carbon	51·55 „ 53·06
Ash	10·80 „ 6·98
Sulphur	1·25 „ 0·96
Coke	63·60 „ 61·00

Dr. A. Simons, of the Ferreira Gold Mining Company of Johannesburg, analysed samples with similar results, and to his report he adds:—"The coal will yield a coke capable of competing with the best imported coke. It is a good household and good steam coal."

It was tested by the Johannesburg Lighting Co. at the request of Dr. Simons. They reported a yield of 11,000 cubic feet of gas and nearly 14 cwts. of coke per ton of coal (2,240 lbs.); the illuminating power of the gas burnt in a standard argand burner, burning 5 feet of gas per hour, was found to vary between 17·51 and 17·54 sperm candles.

A train weighing altogether 119 tons ran from Durban to Botha's Hill station, 31¼ miles, reaching an altitude of 2,425 feet, the steepest gradients being 1 in 30, in 2 hours 41 minutes, on a consumption of 17 cwts. of coal. No difficulty was experienced during the run in keeping the steam pressure at 130 lbs., the engine reaching its destination under full pressure, and it could have run 6 miles more. Finally, on cleaning the fire-box there was very little clinker, and scarcely any ashes in the smoke-box.

The mineral rights are held under the Corporation of Newcastle for twenty-one years at a yearly rental of 5s. per acre, and a royalty of 3d. per ton of coal raised. The company is endeavouring conjointly with the Newcastle Colliery Co. to have their leases extended, and otherwise modified in terms of the regulations for mining on Crown lands; this means that the royalty mitigating the rent, the royalty of 3d. per ton raised will finally be only chargeable against the coal. The Newcastle Corporation offering no opposition, it is only a question of legal *modus operandi*, subject to the sanction of the Government (which has now been obtained).

The colliery is $1\frac{3}{4}$ miles distant from the Natal Government Railway. The branch line over an easy country has been surveyed, necessitating only 9,775 cubic yards of excavation, earth and banks, and 124 yards of concrete and masonry.

(2) *The Newcastle Colliery.*—The company possess three blocks N, O, P, of the Newcastle town lands.

Fig. 4, Plate XXIV., shows the configuration and area of the property, the position of boreholes, and of the adit by which the seam is now worked. The following is a detailed section of the working seam :—

Sandstone-roof, hard and firm.	Ft.	Ins.
Coal	2	6
Splint-band...	0	6
Coal	0	4
Band, variable	0	3
Coal	1	0
Band, variable	0	3
Coal	1	2
Total thickness	6	0

Judging from the contour of the surface, and the lie of the seam, coal extends over the whole area of the property, and deducting this from the area already worked, there remains 270 acres of coal of an average thickness of 4 feet 3 inches, containing, say, 1,814,940 tons of available coal, of which one-fourth will be nuts and dust, and three-fourths will be round coal.

It is extremely probable that the two seams will be found existing on the Southern Cross property, but in the absence of sufficient boreholes and surface-levelling, it is only a possibility; therefore the estimate of available coal in this estate is limited to the tonnage given above.

The remarks as to the quality of the coal of the Southern Cross colliery equally apply to the coal of the Newcastle colliery, both being the best Natal coal.

This company has a nominal capital of £20,000, of which £12,800 is paid up. The mineral rights of the estate is held for twenty-one years, from the Corporation of Newcastle, at a yearly rental of 12s. 8d. per acre, and a royalty of 3d. per ton of coal raised. The mine is connected to the Natal Government Railway by a 2 feet gauge light tramway of portable type. The present output is about 40 tons daily.

Several blocks of the Newcastle town lands have been marked by the corporation on the assumption that the Southern Cross and Newcastle colliery seams are represented therein, but a portion only, however, will be coal-bearing.

Lennoxton Mines.—These mines are situate north-west of Newcastle, but both of them are now closed, and are too far removed from the Natal Government Railway to be of importance in the near future.

Braakfontein Farm.—On this farm, on the right bank of the Ingagane river, lies the same coal-seam as that found in the Newcastle town lands, on the opposite bank, the coal cropping out on both sides. The farm is unproved, but all the indications, however, are most favourable.

Rooi Pont Farm.—The area of this farm is 3,043 acres. Its position is shown on Fig. 3, Plate XXIV. The eastern half of the farm is much cut up by kopjes (or craggy hillocks), having very steep sides. The summits form elevated plateaus covered with good grass. From careful observation the writer came to the conclusion that the portion of the farm lying east of the Natal Government Railway (low-lying land) must be regarded as containing no coal, which leaves 1,650 acres; over the greater part, if not the whole of this area, the seam will be found to exist.

The seam has been proved by a shaft sunk to a depth of 28 feet on the side of the kopje, and by an adit. A section, east and west, across the farm is shown in Fig. 5, Plate XXIV.

An average section of the seam is as follows:—

	Ft.	Ins.
Sandstone	——	
Shale	——	
Coal, bright and cubical	2	0
Band, with strings of coal	0	5
Splint-band...	0	5
Coal, bright and cubical	0	6
Band	0	3
Coal, bright	0	4
Band	0	2½
Coal	1	0
Total thickness	5	1½

Taking the average coal section at 3 feet 8 inches, there will be 6,016,000 tons of workable coal. The coal described as bright and cubical is a glossy friable coal, splitting up into small cubes, and more of a bituminous than an anthracitic nature.

The property is owned, and was being worked when the writer was in Natal, by Mr. Draper, an old colonist.

Tiger Kloof Farm.—This farm (Fig. 6, Plate XXIV.,) had originally an area of 3,978 acres, but the Natal Government Railway cutting through it reduced it to 3,953 acres. There are sufficient indications of coal to

justify the expense of proving it systematically, which is now being done. A shaft on the northern boundary of the farm, by which the seam was for some time, twelve years ago, worked for local sale, has now fallen in; but, according to a miner, it showed a section of 5 feet of coal.

On the south-eastern corner of the farm is an outcrop through which the Natal Government Railway passes. The existence of sections of 5 feet of coal in the northern and southern extremities of the eastern portion of the farm, coupled with the fact that the surface rises toward the kopjes on the west, is a fair proof of the existence of coal on that portion of the farm from the railway to the western boundary.

The kopjes being formed by an outflow of trap rock that can be traced to a point of eruption on the Natal side of the Drakensberg, it is not probable that the coal underneath has suffered any serious disturbance. Coal, then, may be said, with sufficient accuracy, to exist on two-thirds of the property with a workable section of 5 feet, and, after deducting for loss in working, containing 20,000,000 tons of available coal. Among the great advantages of this property is its accessibility to the Natal Government Railway, requiring nothing more than a short siding, and the fact that the coal can be approached and mined by adits.

The Ballengeich Farm.—Fig. 3, Plate XXIV., shows the relative positions of this farm, and of the Natal Government Railway, with the Ingagane river between them.

Fig. 7, Plate XXIV., shows the Horn river, an impetuous torrent, often flooded in the rainy season, flowing nearly at right angles to the Ingagane river, the consequence being that when the Horn river is flooded it dams the Ingagane river and causes it to overflow its banks, the flood covering the lowland on the left bank, across which the branch line would have to be constructed. A substantial viaduct, 600 feet long, would be required, entailing an outlay of £18,000 for bridge and railway. The connecting of the contemplated colliery to the railway could be accomplished at a smaller original cost by a rope (aërial) tramway, but such a mode of transport would unavoidably be attended by contingencies that might ultimately prove very serious.

The outcrops of coal are denoted by the letters *a*, *b*, *c*, *d*, *e*, and *f* (Fig. 7, Plate XXIV.). At none of these points is the seam shown in its full height, as all the outcrops are injured by exposure to atmospheric agencies, and a reliable section could not be obtained without excavating for some distance. Mr. North made an excavation at the point *e* as follows:—

	Ft.	Ins.
Hard coal	1	0
Grey shale	0	3
Coal, hard	3	8
Arenaceous shale	0	6
Dark grey sandstone, very hard	0	6
Coal	0	5
Shale and coal	0	6
Coal	0	8
Parting		
Coal	0	8
Parting		
Coal, hard and good	2	1
Sandstone	0	3
Coal, good	1	0
Parting, with iron pyrites	0	1½
Coal, good	1	2
Total thickness ...	12	9½

Although this section shows a considerable thickness of coal, it at the same time shows a large number of bands, a very serious item, when the working and screening of the coal is considered. This may be taken as an average section of the main seam over the farm.

The outcrop at *d* is of two small adjacent seams, lying at some height above the main seam, and shown in section of No. 3 borehole (Appendix C); these, however, owing to their thinness, are, of course, of no value, except as evidence of the regularity of the strata. The strata overlying the main seam are far more regular, as regards character and thickness, than was observed on other farms in the neighbourhood.

Boreholes, Nos. 5, 6, and 7, were put down on this farm. No. 7 hole, drilled near the shaft, went 90 feet down, coal was found at 48 to 60 feet and was analysed. In No. 5 hole, drilled near the Kafir kraals, coal was found at 41 feet 6 inches, then 6 inches shale-band, 52 feet to 55 feet coal again, the roof being shale and the floor sandstone.

The other boreholes, one of which was put down near Mr. Degenaar's house, struck whinstone, whether a dyke or outflows from a dyke is unknown.

The shaft is 43 feet deep, and is sunk through the seam, where, according to Mr North, there is 9 feet of coal.

No. 1 borehole (Appendix C) is situate on the lowest ground, but it did not strike the seam, which had been denuded, in all probability, by the action of the Horn river. But, owing to the heavy dip of the seam to the south, the seam was struck in the No. 2 borehole (see Appendix C); the heavy dip may be due to faulting, which cannot be determined until underground development has proceeded somewhat. There is no external

evidence of faulting, and the difference of level of the seam in the shaft at the Horn river, and at Nos. 2 and 3 boreholes (Appendix C), is most probably due to dip alone; still, there may, however, be some small faults. Unluckily, the No. 4 borehole struck a whin dyke.

The order of succession of strata may be taken as follows:—

(*a*) Basaltic overlying formation, forming dykes and outflows, piercing and capping the kopjes.

(*b*) Laminated grey rock and thin bands of shale, and thin local seam or seams of coal.

(*c*) False-bedded sandstones.

(*d*) Laminated, arenaceous, micaceous, black and grey shales, with sometimes a band of sandstone forming the roof of the seam.

(*e*) Main coal-seam.

(*f*) Hard sandstone-floor.

The roof of the seam is in most cases composed of shale, which means an increased cost of working, and is very likely to be water-bearing.

In the section of the seam given by Mr. North, the coal is much more split up by bands than it is in the sections given by the boreholes, which makes it probable that the small bands of shaly matter have been passed through whilst drilling in the coal, and washed away (if there was any shale core) by the water pumped down to keep the diamond crown cool and the hole clean.

The tops and sides of nearly all the hills are covered with large whin (basaltic) boulders; and in places the bedded rock is visible. These kopjes or hills owe their existence and origin to this protecting cap of doleritic rock, which has shielded the underlying strata from the various denuding agents that have at one time carried away the original surface. This whinstone exists in two conditions or forms: (*a*) as dykes and (*b*) as outflows. (See Fig. 8, Plate XXIV.)

A considerable part of the hill-surface is formed by overflow whinstone, the dykes existing in a truncated form. It is very probable therefore that the seam will exist for a considerable distance beneath the hills, but whoever works the coal must expect to contend with a considerable number of dykes, which exist also in the lower part, but unaccompanied by outflows. The coal will be more or less indurated in the vicinity of these intrusive masses, and probably useless and unsaleable.

After allowing a liberal deduction for loss by denudation and dykes, there may be 2,800 (out of the 4,005) acres of workable coal, containing a total of 51,010,400 tons of workable coal and at least 38,257,800 tons of vendable round coal.

Mr. C. Hall wrote, in 1889, with reference to the coal from samples of cores from Nos. 5 and 6 boreholes:—"The coal is of fair quality, part anthracitic and part clear-burning bituminous coal, the latter with 22 per cent. of volatile matter and 14 per cent. of ash. It does not light easily, but once started continues to burn, giving out a steady heat as long as there is a good draught. The clear-burning part cakes slightly, but does not form clinker. The ash remains in friable lumps, retaining the original form of the coal." He considered, as far as he could judge from a small sample, that it was better suited for steam raising than for household purposes. The quality of the coal from No. 5 hole, so far as percentage of ash was concerned, was slightly better than the more solid coal.

Mr. E. Neville, Government chemist, wrote in 1891, respecting a sample of coal taken from a borehole on the Ballengeich farm. "It is a hard, compact, dull, greyish-black coal, with a specific gravity of 1·499, of the following composition:—

	Per Cent.
Moisture	1·68
Volatile hydrocarbons	10·84
Coke, free from ash	71·32
Sulphur	0·88
Ash	15·28

The coal is a semi-anthracite of good quality, containing a considerable percentage of ash, and yielding an amorphous coke. Mixed with a more bituminous coal and burnt with a good draught, it would be an economical coal for raising steam. Its small proportion of sulphur and the absence of phosphorus render it suitable for most metallurgical processes."

The Leicester Farm.—This farm, with an area of 3,956 acres, is situated in a favourable position on the right bank of the Ingagane river, the Natal Government Railway running through the estate for about 2½ miles (Fig. 3, Plate XXIV.). This property is not actually proved otherwise than by an out-crop on the right bank of the Ingagane river, close to the railway, and a trial shaft on the opposite side of the line. According to Mr. North, there is a seam of 8 feet 9 inches in thickness. In the absence of a detailed section, and judging from the coal lying at the mouth of the shaft, it is difficult to believe that he found the seam to have a clean section. On the contrary, judging from the samples lying at the surface, it is a hard splinty coal, showing a considerably less section of available coal. It is a hard, semi anthracitic coal, which ignited with difficulty, but gave out great heat when burning.

Lately the Commissioner of Mines drilled a borehole (Fig. 7, Plate XXIV.) which went to a depth of 324 feet without passing through any

coal-seam. The coal-bearing portion of this property must therefore lie south of the shaft and the outcrop, the contour of which is favourable to its existence.

Railways.

Over 375 miles of railways are open for traffic. The main trunk line extends from Durban to Ladysmith (189¼ miles), Biggarsberg and Newcastle (268 miles), and is now completed northward to Charlestown (304 miles) on the Transvaal frontier. There is a branch line from Biggarsberg to the Dundee coal-fields (which will ultimately prove the main line to Zululand) and from Ladysmith to Harrismith *via* Van Reenan's Pass into the Orange Free State. From Durban lines run northward to Verulam (19 miles) and southward to Isipingo (7 miles).

The railways are the property of the Natal Government. The greater part is laid over very difficult country, especially between Durban and Pietermaritzburg, and over the Town Hill, north of the latter town, and those parts which cross the Berg at Van Reenan's Pass, and Langs Nek below Amajuba.

The distance by wagon-road from Durban to Pietermaritzburg is 65 miles, and about 70¼ miles by rail. Some of the curves are extremely sharp. The steepest gradient is about 1 in 35.

The following figures will show the excellent management of this railway system :—

1886, 189½	miles open ;	revenue,	£108,878.
1887, 216¾	„	„	257,877.
1888, 220	„	„	349,184.
1889, 241	„	„	535,261.
1890, 286	„	„	606,713.

For the financial year, ending June 30th, 1891 (taking the first six months of the year as a basis of calculation), the following results were arrived at :—

Gross income	£587,499
Gross expenditure	£356,957
Excess income	£230,542
Earnings per open mile	£1,882
Earnings per train mile	14s. 2·74d.
Expenditure per open mile	£1,144
Expenditure per train mile	8s. 7·72d.
Passenger earnings	£111,701
Goods earnings	£455,730
Miscellaneous earnings	£20,060
Interest on capital	£235,360
Proportion of expenses to earnings...	...60 per cent.

The proportion borne by working expenditure to receipts in some other countries may be usefully recalled by way of comparison :—In Great Britain it is 53 per cent.; in Cape Colony, 55 per cent.; in Germany, 52½ per cent.; in Austria, 54 per cent. for State lines, and 44¾ per cent. for companies' lines; in Belgium, 62 and 52½ per cent. for State and companies' lines; and in France, 90 and 50 per cent. for State and companies' lines.

The Harbour.

Durban is the only seaport of Natal, with a population of 25,000, of which one-quarter are natives and Indians. The business of the harbour has in past years been much impeded by a shifting bar of sand across the mouth of the channel.

The average depth of the channel at low water has, however, increased from 6 feet 1 inch in 1882 to 10 feet 9 inches in 1891, and the harbour works are still being continued at considerable cost to the Colony. The total sums paid out of the Treasury up to May, 1890, and used by the Harbour Board, amounted to £640,729.

The maximum draught brought inside in 1890 was 16 feet 8 inches and the mean draught 13 feet 8 inches. 118 steamers came inside during 1890 as compared with 114 in 1889, 68 in 1888, and 63 in 1887; 179 sailing vessels came inside during 1890 as compared with 196 in 1889, 165 in 1888, and 135 in 1887; the total increase in tonnage which has arrived at the port in 1890 was 11,937 over 1889, 133,327 over 1888, and 258,095 net tons over 1887.

There are as yet no proper facilities for shipping coal. The cost of coaling vessels is about as follows :—

	Per Ton. s.	d.
I.—Coaling alongside the wharf :—		
(*a*) When shoots can be applied	1	0
(*b*) When the steam-crane is used	1	6
(*c*) High boats, where baskets have to be adopted	2	0
(*d*) Sacking and trimming	0	6

	Per Ton. s.	d.		
II.—Coaling outside :—				
Putting into lighters	1	0		
Taking out of lighters	5	0		
Putting under hatches	1	6		
			7	6

The authorities have, however, under consideration the erection of hydraulic tipping-appliances, designed by Sir W. G. Armstrong, Mitchell, & Co., to suit the 20 tons trucks of the Natal Government Railway, which will greatly reduce the cost of coaling.

Present and Possible Future State of the Coal Trade.

The Natal coal industry is of course extremely young—in fact it may be regarded as not more than four years old. The unavoidably limited demand at the beginning of the industry has been a check on its development, but it now exceeds the supply.

On that question, as well as that of transport, His Excellency the Governor expressed himself at the opening of the Legislative Council on the 5th of May, 1892, as follows:—"The supply of coal for export is assuming such proportions that it will probably be necessary ere long to take into consideration the possibility of increasing the carrying capacity of our railway."

There are so many stations on the Indian Ocean where a considerable tonnage of coal is actually supplied at higher prices, and in no better quality than procurable from Natal, and where it will be permanently required, that it is a natural inference to say, as above, that the demand for Natal coal will be limited only by the ability of their collieries to supply it.

The present import duty on all coal or coke entering the Transvaal, viz., 15s. per 100 lbs. weight, which practically prohibits the introduction of coal into that country, where large quantities of local and inferior coal are used at the gold and silver mines, and for gas-making at Johannesburg. But with the extension of the Natal railways, which is now being negotiated, and the formation of a Customs Union of all the South African States, there can be no doubt that this present onerous duty will be abolished; such an abolition would enormously increase the output of coal, and give a great impetus to trade generally.

The Natal Government Railways are using local coal, thereby effecting a very considerable saving in working expenses.

The following are the quantities supplied by various collieries to the railway for 18 months ending November 26th, 1889:—

	Tons.
Elands Laagte colliery	9,561
Still's ,, ...	638
Ramsay's ,, ...	5,320
Wallsend ,, ...	974
Dundee ,, ...	37,112
Acol ,, ...	12
Lennoxton ,, ...	2,349
Total	56,016

The coal is used for domestic and for manufacturing purposes. It is also supplied to many steamers in the bay at Durban at about 20s. per

ton. During the year ending June 30th, 1890, 548 tons were exported; 1891, 19,923 tons; and in 1892, 30,000 tons were absorbed for bunkering and export purposes. In 1890, 81,000 tons of coal were raised, or more than double the output of 1889; in 1891, 93,551 tons; and in 1892, 118,932 tons.

There appears to be no reason why the bunkering trade alone should not exceed 80,000 tons per annum. For an export trade, however, Natal will have to compete with English coal at the Cape ports and with Australian coal at the Mauritius. The possibilities of the case, however, are undoubtedly in favour of Natal coal being sold at these ports. The Pungwe river (recently established) and Delagoa Bay are possible markets, but the demand at the latter place, however, amounts only to 4,000 tons.

The price at the pit's mouth has greatly fallen during the last two years, and is now under 10s. per ton. A director of one of the principal coal companies has stated that the cost of raising coal during one month (September, 1890) was 6s. 8d. per ton of round coal, which has been reduced since then.

The Natal Government Railway conveys coal at a fixed rate of ½d. per ton per mile, with a minimum rate of 2s. per ton. All coke and briquette fuel entering the colony is chargeable with duty at the rate of 3s. per ton. Mining and other machinery is admitted free of duty and conveyed at the lowest goods rate, viz., £3 5s. and £5 per ton respectively.

Climate.

The climate of Natal varies considerably, and although it is nearer the tropics than Cape Colony, its mean annual temperature is scarcely higher.

At Durban the highest temperature in the shade in September, 1890, was 110 degs. Fahr., which was very exceptional; the lowest being 47 degs. in July. In Newcastle the highest temperature in the shade last summer (at Christmas) was 96 degs., and the lowest showed about 10 degs. of frost in July. The winter is bright and dry, and cold at night in the uplands; while the summer heat is tempered by a cloudy sky, and almost daily thunderstorms and heavy rains.

Population.

By the census of 1891 the population was returned as:—Europeans, mainly British, 46,788; Indians, 41,142; natives, 455,983.

Labour.

Both coolie and native labour is used in mining operations, opinions differing as to which is the best. The native labour is very unreliable, as

the "boys" will not remain at their work more than a few months at a time, though, if well treated, they return again and again. Coolies, on the other hand, can be indentured for a fixed period, but such as have as yet been introduced into the colony have not proved so successful in the mines as on the railroads or sugar-cane fields.

The writer is of opinion that the right caste has not hitherto been introduced for mining work. Particulars as to wages, etc., have already been given under the heading of the Ramsay mine and Dundee colliery. White labour is scarce and very dear.

The price of mealies (Indian corn), the chief article of food with the natives, varies very greatly. The cost of feeding natives usually averages about 10s. per head per month, but in 1889, on account of the famine price of mealies, it cost, in Durban, about £1 10s. a head per month.

Timber.

Natal cannot be considered a well-wooded country, for with the exception of the coast-lands and in the kloofs of the Drakensberg range, the country is very bare in this respect. Arboriculture, however, has of late years been taken up to some extent, and certain kinds of trees are found to grow very well.

The trees most planted are pines *(Pinus insignis)* being most suited for mine timber, as well as being ornamental; black wattle *(Acacia decurrens*, variety *mollissima)* on account of its bark, used for tanning purposes, and gum trees, especially the blue gum *(Eucalyptus globulus)*. The rate of growth of these latter trees is most remarkable (see Appendix E).

Owing to the strong character of the roof, in no mine that the writer visited did he see many props. In fact, in the Dundee and Newcastle mines hardly a prop was used.

APPENDIX A.

In pursuance of the request of the Commissioner of Mines, the writer visited an alleged discovery of oil-shale and coal, cropping out on the side of one of the hills forming the Marotshiya ridge (Figs. 9 and 10, Plate XXIV.) between the Umkomanzi and Lotini rivers, in the county of Pietermaritzsburg, in January, 1892. This turned out to be a black-and-grey carbonaceous shale-bed, about 19 feet thick, but hardly an oil-shale; layers of bright shining coal, about half an inch thick, were interstratified with the shale.

Geologically, the discovery of this carbonaceous stratum was of great interest, as proof that in all probability coal of a workable thickness existed in the district. Owing to the extreme variableness of the coal-measures of the colony it is quite possible that this carbonaceous bed may at some other point contain considerably

more coal than at Antel's discovery, and in a more regularly stratified condition. The writer regrets that he was unable to devote time enough to visit Mileman's discovery, where he was informed the coal was much more developed than at Antel's.

It may be worthy of remark to state that the base of the coal-measures in the Impendhla district is at least 1,000 feet above the base in the Newcastle district. Commencing at Pietermaritzburg (where the altitude is 2,218·4 feet) we are on the Pietermaritzburg Shales; near the summit of Edendale Hill the upper beds of the series are found, characterized by the bright red and variegated colour of the shales and by beds of light-coloured concretionary sandstone, which latter presents very curious weathering effects.

The office of the administrator of native law in the Impendhla location is situate at about the base of the coal-measures, the altitude being (by aneroid) 5,190 feet or 2,810 feet above Pietermaritzburg. The coal-measures, which were discovered for the first time during the journey at the Impendhla, are easily located by their horizontal position and difference in colour from the lower rocks. Although the Pietermaritzburg Shale series was here and there traversed by dioritic dykes, it was very noticeable that on approaching the coal-measures and nearing the Drakensberg range, these became more numerous and so plainly visible that the writer was able to trace some of them for several miles.

The writer made a rough section of the Impendhla hill, which is shown in Fig. 11, along the line C D, in Fig. 9, Plate XXIV.

APPENDIX B.

Sections of Boreholes, etc., made by the Southern Cross Coal Mining Syndicate (Fig. 4, Plate XXIV.).

a.—Borehole.

Description of Strata.	Thickness of Strata. Ft.	In.	Depth from Surface. Ft.	In.
Clay	3	0	3	0
Sandstone	29	8	32	8
COAL	0	2	32	10
Sandstone	9	0	41	10
COAL	0	0½	41	10½
Sandstone	8	2	50	0½
COAL	0	3	50	3½
Sandstone	4	0	54	3½

b.—Borehole.

Description of Strata.	Thickness of Strata. Ft.	In.	Depth from Surface. Ft.	In.
Clay	3	0	3	0
Ironstone	1	0	4	0
Sandstone	14	8	18	8
Sandstone	10	3	28	11
Shale...	8	9	37	8
Sandstone	10	0	47	8
Shale...	9	5	57	1
Sandstone	20	3	77	4
COAL	0	8	78	0
Sandstone	9	2	87	2
COAL	0	9	87	11
Ironstone	9	5	97	4

c.—Borehole.

Description of Strata.	Thickness of Strata. Ft.	In.	Depth from Surface. Ft.	In.
Clay	2	8	2	8
Ironstone	1	7	4	3
Sandstone	5	9	10	0
Sandstone	17	10	27	10

c.—Borehole.—*Continued*.

Description of Strata.	Thickness of Strata. Ft.	In.	Depth from Surface. Ft.	In.
Shale	0	7	28	5
Sandstone	10	0	38	5
COAL	6	0	44	5
Sandstone	25	5	69	10
COAL	2	1	71	11
Shale	0	4	72	3
Sandstone	16	0	88	3

d.—Borehole.

Description of Strata.	Thickness of Strata. Ft.	In.	Depth from Surface. Ft.	In.
Clay	15	0	15	0
Sandstone	10	4	25	4
Shale	8	3	33	7
Sandstone	0	4	33	11
COAL	7	9	41	8
Sandstone	22	10	64	6
Shale	2	8	67	2
Sandstone	0	6	67	8
Shale	0	5	68	1
Sandstone	7	4	75	5
COAL	2	5	77	10
Shale	0	2	78	0
COAL	0	5	78	5
Sandstone	1	2	79	7
COAL	2	1	81	8
Shale	2	11	84	7
Sandstone	33	0	117	7
COAL	0	10	118	5

d.—BOREHOLE.—*Continued.*

Description of Strata.	Thickness of Strata. Ft.	In.	Depth from Surface. Ft.	In.
Shale	3	0	121	5
COAL	0	2	121	7
Sandstone	9	10	131	5

e.—SHAFT.

Timbered	19	0	19	0
Sandstone	19	6	38	6
Shale	10	9	49	3
COAL	2	0	51	3
Shale parting ...	0	3	51	6
COAL	2	9	54	3

f.—BOREHOLE.

Description of Strata.	Thickness of Strata. Ft.	In.	Depth from Surface. Ft.	In.
Clay	2	0	2	0
Ironstone	10	2	12	2
Sandstone	34	6	46	8
COAL	5	7	52	3
Sandstone	33	5	85	8
COAL	1	7	87	3
Shale	12	9	100	0
Sandstone	100	0	200	0

APPENDIX C.

SECTIONS OF BOREHOLES MADE ON THE BALLENGEICH FARM (Fig. 7, Plate XXIV.).

No. 1.—BOREHOLE.

Description of Strata.	Thickness of Strata. Ft.	In.	Depth from Surface. Ft.	In.
Sandy clay	24	5	24	5
Ironstone grit ...	0	2	24	7
Sandstone	2	5	27	0
Black band	0	7	27	7
Sandstone	12	5	40	0

No. 2.—BOREHOLE.

Soil	9	0	9	0
Sandstone	55	1	64	1
Shale	6	1½	70	2½
Sandstone	3	0	73	2½
COAL	9	7	82	9½
Black splint band ...	0	4	83	1½
COAL	2	7½	85	9
Sandstone	18	0	103	9

No. 3.—BOREHOLE.

Clay	13	0	13	0
Sandstone	42	7½	55	7½
COAL	0	5	56	0½

No. 3 BOREHOLE.—*Continued.*

Description of Strata.	Thickness of Strata. Ft.	In.	Depth from Surface. Ft.	In.
Sandstone	0	8¼	56	8¾
COAL	0	3	56	11¾
Sandstone	37	8	94	7¾
Shale	3	9	98	4¾
Sandstone	5	5	103	9¾
Shale	14	6	118	3¾
COAL	3	7	121	10¾
Black band	0	1	121	11¾
COAL	0	7	122	6¾
Black band	0	8	123	2¾
COAL	5	4	128	6¾
Black band	0	2	128	8¾
COAL	2	11¼	131	8
Sandstone	0	4	132	0

No. 4.—BOREHOLE.

Soil	20	0	20	0
Sandstone	9	10½	29	10½
Whin	0	3	30	1½

APPENDIX D.

JOURNAL OF BORINGS ON THE BALLENGEICH FARM.

No. 1 BOREHOLE.

Date. 1891.	Bored in each Bed. Ft.	In.	Thickness of Bed. Ft.	In.	Total Depth. Ft.	In.	Core obtained. Ft.	In.	Nature of Strata.	Remarks.
Nov. 16	16	6	...		16	6	Nil.		Sandy clay ...	Waiting for pump casting.
„ 18	7	11	24	5	24	5	„		Do. ...	Received new pump and fitted up same.
	0	2	0	2	24	7	„		Ironstone grit.	
	2	5	2	5	27	0	„		Sandstone.	
	0	7	0	7	27	7	„		Black shale.	
	7	6	7	6	35	1	1	3	Sandstone.	
„ 19	4	11	...		40	0	0	6	Coarse sandstone grit, with black parting.	Finished boring, dismantled drill.

No. 2 Borehole.

Date. 1891.	Bored in each Bed.	Thickness of Bed.	Total Depth.	Core obtained.	Nature of Strata.	Remarks.
	Ft. In.	Ft. In.	Ft. In	Ft. In.		
Nov. 20	...	...	...	...	... ...	Shifting plant and erecting same at No. 2 site.
„ 21	0 10	0 10	0 10	Nil.	Black surface soil.	
	2 8	2 8	3 6	„	Yellow sandy soil.	
	1 6	1 6	5 0	„	Red ferruginous soil.	
	4 0	4 0	9 0	„	Stiff yellow clay.	
„ 23	4 0	4 0	13 0	„	Sandstone	Removing tents to No. 2 site.
	0 4	0 4	13 4	„	Dark sandstone.	
„ 24	1 6	1 6	14 10	...	Coarse white sandstone.	
	0 1	0 11	14 11	...	Very hard, fine, dark-brown sandstone.	
	11 7	11 7	26 6	...	Coarse white sandstone.	
	6 0	...	32 6	4 0	Hard grit sandstone.	
„ 25	15 1	...	47 7	11 6	Do. do.	
„ 26	16 6	37 7	64 1	4 3	Do. do.	
	2 6	...	66 7	1 4	Black, micaceous, and arenaceous shales.	
„ 27	3 7½	6 1½	70 2½	4 4	Do. do. ...	Core, part of yesterday's boring as well as to-day's.
	3 0	3 0	73 2½	...	White sandstone, hard.	
	8 1½	...	81 4	...	Coal, cubical near top, and bright.	
„ 28	1 5½	9 7	82 9½	...	Do. do.	
	0 4	0 4	83 1½	...	Black splint band.	
	2 7½	2 7½	85 9	1 0	Coal, anthracitic and slaty looking.	
	0 7½	...	86 4½	0 6½	Hard grey sandstone with pyrites.	
„ 30	...	...	...	...	... ...	Kaffirs carrying fire-wood during morning, and stopped by heavy rains rest of day.
Dec. 1	10 2½	...	96 7	3 1	Same as last, but free from pyrites	Lost 5 barrels of water in borehole, and stopped by heavy rain towards close of the day.
„ 2	7 2	...	103 9	10 0	Same as last	Core of yesterday's boring as well as to day's. Lost 5 barrels of water down borehole.

No. 3 Borehole.

Date. 1891.	Bored in each Bed.	Thickness of Bed.	Total Depth.	Core obtained.	Nature of Strata.	Remarks.
	Ft. In.	Ft. In.	Ft. In.	Ft. In.		
Dec. 3, 4, & 5	...	...	...	...	... ...	Dismantling, shifting, and erecting drill and tents at No. 3 site.
„ 7 ...	13 0	13 0	13 0	Nil.	Stiff clay.	
„ 8 ...	16 4	16 4	29 4	2 4	Very close-grained sandstone, with bands of arenaceous, carbonaceous, and micaceous shale.	
	6 0	...	35 4	1 8	Coarse-grained sandstone, very hard.	
„ 9 ...	2 0	8 0	37 4	4 8	Do. do.	
	11 2½	11 2½	48 6½	6 4	Very close-grained sandstone.	
	1 0	1 0	49 6½	1 0	Sandstone, with thin bands of micaceous shale intermixed.	
„ 10 ...	2 3	2 3	51 9½	2 3	Sandstone, grey and hard.	
	3 10	3 10	55 7½	3 10	Very close-grained sandstone, with innumerable wavy bands of black micaceous shale, false-bedded.	
	0 5	0 5	56 0½	0 3	Coal, hard and bright, with thin dull bands.	
	0 8¼	0 8¼	56 8¾	0 8¼	Sandstone, with layers of arenaceous shale.	
	0 3	0 3	56 11¾	0 1	Coal, bright and anthracitic.	
	4 2	4 2	61 1¾	2 9½	Sandstone, with micaceous shale.	
„ 11 ...	5 2	...	66 3¾	2 6	Coarse white sandstone	Lost 36 barrels of water in borehole; shows breaks in sandstone.
„ 12 ...	2 0	...	68 3¾	2 0	Do. do. very hard	Do. 14 do. do.
„ 14 ...	6 7	...	74 10¾	0 8	Do. do.	Lost core in borehole and 20 barrels of water.
„ 15 ...	6 7¼	...	81 6	4 2 5 2	Do. do.	Found lost core; stopped half-day by heavy rain.
„ 16 ...	5 8¾	...	87 2¾	4 0	Do. do.	Stopped half-day; Kafirs carrying mealie-meal.
„ 17 ...	6 5	...	93 7¾	3 0	Do. do.	

No. 3 Borehole.—*Continued.*

Date. 1891.	Bored in each Bed.	Thickness of Bed.	Total Depth.	Core obtained.	Nature of Strata.	Remarks.
	Ft. In.	Ft. In.	Ft. In.	Ft. In.		
Dec. 18 ...	1 0	33 6	94 $7\frac{3}{4}$	0 8	Coarse white sandstone.	
	3 9	3 9	98 $4\frac{3}{4}$	2 0	Black arenaceous shale.	
,, 19 ...	1 5	1 5	99 $9\frac{3}{4}$	1 5	Coarse white sandstone, very hard	Stopped for an hour by rain.
,, 21 ...	2 11	...	102 $8\frac{3}{4}$	1 1	Fine-grained sandstone	Lost part of core in borehole; stopped one hour repairing pump.
,, 22 ...	1 1	4 0	103 $9\frac{3}{4}$	1 3	Do. do.	Recovered lost core of yesterday.
				1 1		
	9 1	...	112 $10\frac{3}{4}$	9 0	Sandstone alternating with bands of black micaceous shale, false-bedded.	
,, 23 ...	5 5	14 6	118 $3\frac{3}{4}$	4 3	Do. do. do.	
	3 7	3 7	121 $10\frac{3}{4}$	2 0	Coal.	
	0 1	0 1	121 $11\frac{3}{4}$	0 1	Black band.	
	0 7	0 7	122 $6\frac{3}{4}$	0 5	Coal.	
	0 4	...	122 $10\frac{3}{4}$	0 3	Black shale.	
,, 24 ...	0 4	0 8	123 $2\frac{3}{4}$	0 3	Do.	
	5 4	5 4	128 $6\frac{3}{4}$	1 0	Coal, bright, glossy, cubical towards bottom.	
	0 2	0 2	128 $8\frac{3}{4}$	0 1	Black band.	
	2 $11\frac{1}{4}$	2 $11\frac{1}{4}$	131 8	2 4	Coal, dark anthracitic, with layers of pyrites.	
	0 4	...	132 0	0 4	Very hard close-grained greyish sandstone.	
No. 4 Borehole.						
Dec. 29	20 0	20 0	20 0	Nil.	Surface soil.	
,, 30	7 0	7 0	27 0	...	Soft sandstone.	
	2 $10\frac{1}{2}$	2 $10\frac{1}{2}$	29 $10\frac{1}{2}$	...	Coarse, gritty sandstone.	
	0 3	...	30 $1\frac{1}{2}$	0 3	Whinstone	It took 5 hours to bore into 3 inches of this igneous mass.

APPENDIX E.

In Mr. H. G. Fourcade's "Report on the Natal Forests" to the Colonial Government, 1889, we find the following particulars as to the growth of gums in the vicinity of Pietermaritzburg:—"Production of wood. In the case of gums the rate of growth may be ascertained accurately, as there are sufficiently old plantations of the kind in Natal. Mr. Topham's plantation near Pietermaritzburg may be taken to be a fairly average one. The oldest plot—40 acres in area—is seventeen years old, and consists of mixed *Eucalyptus globulus*, *Eu. longifolia*, and *Eu. rostrata*, the rate of growth of each appearing not to differ widely. The trees are planted 12 feet apart, so that there are 302 to the acre; they average 4 feet 10 inches in girth at 3 feet from the ground, and 115 feet in height . . . the contents of each tree average 96·3 cubic feet and standing stock per acre 29,083 cubic feet, representing a yearly increment of 1,711 cubic feet per acre. The weight of a cubic foot of *Eu. globulus* (dry wood with bark) averages 40lbs." To Mr. Topham, who has made a study of the subject for many years, the writer is indebted for much general information relating to arboriculture in the colony.

Individual trees of *Pinus insignis*, growing near Howick, show an average yearly increase of 6 inches in girth, attaining a height of 45 feet in ten or twelve years.

APPENDIX F.

When boring on the Walmsley farm (Fig. 3, Plate XXIV.), all the holes were drilled by contract, a hand-worked diamond drilling machine being employed in this as in all other borings. The price paid was 10s. per foot for the first 100 feet, and 15s. per foot for the second 100 feet; all loss of diamonds, break down of machinery, etc., being borne by the contractor.

Mr. Clayton, the contractor, informed the writer that he had obtained the following results in the Newcastle district:—

Holes drilled, 11.
Extreme distance apart, 5 miles.
Total length drilled, 1,144 feet.
Total working days, 50.
Total working hours per day, 10·5.
Average depth drilled per day, 22·88 feet.

The only labour used was Zulu or Swazi-Kafir.

When boring on Ballengeich farm, the writer had the use of the Government drills on paying labour and upkeep. The cost per foot was 3s. 6d., but in this case he personally superintended everything. All the holes were $1\frac{1}{8}$ inches in diameter.

APPENDIX G.

Report upon Natal Coals, by E. Nevill, F.I.C., F.C.S., etc., etc., Government Chemist and Official Assayer for Natal.

I.—Scheme of Analysis.

The technical examination of a coal may be divided into five sections, as follows:—(*a*) The general analysis; (*b*) the quantitative determination of the principal constituents; (*c*) the examination of the physical character; (*d*) the determination of the efficiency as fuel; (*e*) the examination of special properties.

(*a*) The first of these five divisions comprises the determination of the amount of:—i., moisture; ii., volatile hydrocarbons; iii., fixed hydrocarbons; iv., sulphur (total); v., ash.

The determination of these five quantities forms the general analysis of the coal, and serves to indicate its character and probable value as a fuel or for manufacturing or for metallurgical purposes.

The moisture usually ranges between 1 and 3 per cent.

The volatile hydrocarbons vary in quantity from under 15 per cent. in anthracite coals to over 40 per cent. in bituminous coals, and form the portion of the coal giving rise to the flame and smoke.

The fixed hydrocarbons, or the coke free from ash, constitutes the portion of the coal from which is derived the greater part of its heating power. In anthracite coals the fixed hydrocarbons usually amount to over 80 per cent. of the coal, whilst in the bituminous coals the amount falls to less than 50 per cent.

The total amount of sulphur present in a coal is one of the most important factors in determining the value of a coal. In the better class of English coals it varies between $\frac{2}{3}$ per cent. and $1\frac{1}{3}$ per cent.

The ash, or mineral constituents, of the coal should not exceed 10 per cent., the better class of English coals having rather less than half this amount.

The sum of these five quantities will be found to exceed 100, this apparent anomaly being due to the fact that the ash found on burning the coal is really larger than the real inorganic constituents of the coal, and includes a portion of the sulphur and carbon independently reckoned under other heads.

(*b*) The second division affords the data for determining the constitution of the coal, and thus ascertaining its true value. It may be subdivided into:—

Organic constituents—vi., carbon; vii., hydrogen; viii., nitrogen and oxygen; ix., sulphur.

The actual efficiency of a coal depends on the amount of carbon and hydrogen that it contains, as it is the combustion of these constituents which generates the heat evolved on burning. They form over 85 per cent. of the best class of coals. The amount of nitrogen and oxygen are of lesser importance, it mainly serving to indicate the character of the coal. The proportion of sulphur which exists in the coal in combination with the organic constituents is important, because this portion of the total amount of sulphur exerts far less injurious effect than that which is present as pyrites.

The sum of these four percentages constitutes the organic or true coal, and the ratio of the fixed to the volatile hydrocarbons serves to indicate the real nature of the coal. In anthracite coals the ratio usually exceeds 4 to 1, whilst in the bituminous coals it is generally less than 2 to 1.

The second sub-division is :—

Inorganic constituents—x., sulphur as pyrites : xi., sulphur as sulphates ; xii., iron as pyrites; xiii., iron as ash; xiv., pyrites (total).

These inorganic constituents are of less importance, the only one possessing special value being the amount of sulphur existing as pyrites, as it is under this form it exerts most injurious effects. The sulphur as sulphates is quite uninjurious. The total amount of pyrites is always slightly greater than the sum of the sulphur and iron existing as pyrites, as the pyrites always contain a small quantity of other constituents.

(*c*) The principal physical data required for the determination of the character and value of a coal consist of :—(a) specific gravity; and (b) hardness; (c) friability; (d) relative cleanliness : (e) co-efficient of deterioration.

The four last of these are measured on an arbitrary centesimal scale, the quality being denoted by some decimal number between 0·00 and 1·00.

The specific gravity gives the weight of a given volume of coal when the weight of an equal volume of water is taken for unity. The greater the specific gravity the larger the weight of coal which can be stored or carried in the same space.

The hardness of a coal is measured on a scale which makes 0·10 represent a soft imperfect coal, and 0·90 denote a hard stone-like coal.

The friability or propensity to break up into small pieces is measured on a scale which makes 0·10 represent a strong stone-like coal, and 0·90 denote a soft crumbling coal, easily broken up in the fingers.

The relative cleanliness is measured on a scale which makes 0·10 represent a coal full of cavities filled with sandy clay and carrying scattered masses of shale, clay, etc., whilst 0·90 denotes a homogeneous clean coal, free from all signs of foreign matter. No coal can be called clean which does not come up to 0·80 on this scale.

The co-efficient of deterioration measures the stability of a coal or its power of resisting the injurious effects of exposure to air, moisture, and variations of climate. On this scale 0·10 represents a coal little altered by long exposure, whilst 0·90 denotes a coal which is quickly spoilt.

(*d*) Under the head of efficiency, there have been placed the results obtained for judging of the value of the coal by practical trial with an experimental boiler and furnace. These results may be divided under the heads :—(1) evaporative power; (2) residue; (3) smoke; (4) character of fire.

Under the head of evaporative power is given the pounds of water converted into steam by one pound of coal, and the number of pounds of coal that can be economically burnt per square foot of fire-grate area. Upon the scale adopted, the evaporative power of the principal coals may be taken as approximately :—

	Lbs.
Anthracite (best) coals	10·5
Best Welsh steam navigation coals ...	10·0
Welsh steam coals	9·5
North country	8·8
Yorkshire hard steam coals	8·3
Scotch steam coals	8·0
Lancashire coals	7·8

With regard to the speed with which a coal should be capable of being burnt with economy and completeness, it should not be less than 18 lbs. of coal per hour per square foot of fire-grate area, and it is desirable that it should be over 20.

The amount of residue left by a coal on burning should not exceed 15 per cent., and it consists usually of about equal parts of clinker and ashes.

The smoke is given on the Admiralty scale, according to which :—

0 = No smoke.
1 = Smoke just visible.
2 = Light brown smoke fully visible.
3 = Brown smoke, sky visible through smoke.
4 = Dark brown smoke, nearly opaque.
5 = Dark brown smoke, opaque.
6 = Black smoke.
7 = Dense black smoke.

The heading, character of fire, needs no explanation.

(*e*) The only point considered under this heading is the coke left by the coal, but so much depends upon the method by which the coke is made, and the examination has been restricted to determining whether or not the coal is capable of giving a true coke.

II.—Description of Walmsley Coal.

A single sample (of about 2 cwts.) from this property.

It was taken from the upper portion of the seam above the separating band of black shale, from near the outcrop, where the coal has become much weathered.

It is a hard compact greyish coal with a silky lustre, breaking with a semi-conchoidal fracture into rhomboidal masses, and containing small cavities filled with sandy clay. The surface of the coal and the cleavage planes were stained yellow from the oxidation of the thin films of grey iron pyrites carried by the coal.

	Per Cent.
(*a*) *General Analysis.*	
Moisture	2·73
Volatile hydrocarbons	29·54
Coke	53·40
Sulphur	1·17
Ash	14·00
Sum	100·84
(*b*) *Constitution.*	
Organic Constituents—	
Carbon	70·91
Hydrogen	4·73
Nitrogen and oxygen	6·35
Sulphur	0·62
Sum (true organic coal)	82·61
Volatile hydrocarbons	35·50
Fixed hydrocarbons	64·50
Ratio	1·81
Inorganic Constituents—	
Sulphur as pyrites	0·49
Do. as sulphates	0·06
Iron as pyrites	0·43
Do. as ash	0·23
Pyrites (total)	0·92
Easily oxidised sulphur	0·08
(*c*) *Physical data.*	
Specific gravity	1·463
Hardness	0·45
Friability	0·40
Relative cleanliness	0·75
Coefficient of deterioration	0·20
(*d*) *Efficiency.*	
Evaporative power—	
Pounds of water evaporated per pound of coal	9·00
Pounds of coal burnt per hour per square foot of grate area	18·00
Residue (in percentages)—	
Clinker	8·45
Ash	9·22

Smoke (in Admiralty scale numbers)—	Per Cent
Maximum	1·40
Mean	0·60
Percentage of time when no smoke ...	80·00

Character of Fire.—Coal lights fairly well, and burns with a little flame to a dull red mass. Burns better under a forced draught, yields a good deal of soft shaly clinker, which does not adhere to the bars, and a quantity of soft grey ash.

(*c*) *Coke.*—Yields a cindery black coke, soft and friable, and of very indifferent quality.

The Chairman said he was glad to be able from a personal acquaintance with the coal-field described to speak of the value of this paper, because at the present time the key to the supply of coal to vessels in that part of the world was no doubt the coal-field found in Natal. The coal-seams were most puzzling, and gave rise to two conflicting and opposite opinions—one suggesting that the seams were as continuous as those of European and American coal-fields; and another of contrary opinion, and he (the Chairman) from his own observation agreed with the latter. It was absolutely necessary to prove the entire area of a coal-field before they could say it was continuous. Another difficulty in this colony was the manner in which they had laid out their railways. Severe gradients and curves had prevented its natural development, and they must now face the cost of duplicating the single line. Probably the cheapest and best way would be to form another railway to the coast. There was further difficulty at the port, ships having to lie off a considerable distance from the coast, everything being loaded over the side. He noticed in the paper that "the authorities have, however, under consideration the erection of hydraulic tipping-appliances designed by Sir W. G. Armstrong, Mitchell, & Co., to suit the 20 tons trucks of the Natal Government Railway, which will greatly reduce the cost of coaling." In his opinion the 20 tons trucks and the shipping arrangements would require reconsideration. The coal industry of Natal might not affect this country materially as a competitor, but it was valuable as a means of coaling English ships in the Indian Ocean, and he considered the extent of the available coal would warrant the energetic development which the colonials were making.

Mr. T. E. Forster said that although he was not acquainted with this particular coal-field, there were several points which appeared to agree with mining experiences in other colonies; for instance, the leaving of small pillars. Had there been any difficulties in consequence of this? In Australia, where the coal-seams appeared to be somewhat the

same, so far as general conditions were concerned, there was difficulty in getting pillars, left many years ago, without, in many cases, bringing on a serious crush. The railway mileage rates compared favourably with other countries of the same kind. With regard to the export trade they might often be able to supply the coal at a cheaper rate to certain stations, where there was a demand for it, but it was often found that such trade did not depend so much on price as on the possibility of obtaining tonnage to carry it. Vessels would always go where best paying freights took them, and consequently it might be difficult to carry on an export trade with sufficient steadiness to suit regular customers.

Mr. W. Cochrane asked Mr. Redmayne to give the quoted prices of Australian and English coals in the competing markets mentioned in his paper?

Mr. John Bigland said the writer described the deposits as aqueous. Did he mean marine or freshwater deposits? They had very few papers with regard to coal-fields in the Triassic system, and it was somewhat new to many of them. He was prepared to believe, after hearing the paper, that it was a marine deposit, and that the coal had been deposited as drift in the same way as Continental geologists asserted.

Mr. M. Walton Brown asked whether the analyses given were the results of average samples of the seam, or from picked portions of the coal separated from stone?

Mr. Redmayne said they were from an average sample, but without band.

Mr. Cochrane—A sample of the marketable coal?

Mr. Redmayne—Yes.

Mr. Redmayne, replying to the discussion, said there was some talk of doubling the railway, but it was regarded as impossible, the bridges being only made for a narrow-gauge ($3\frac{1}{2}$ feet) railway, and it would be found cheaper to make an independent railway to Durban. Subsidences had taken place owing to attempts made to remove the thin pillars; he had been asked and had recommended certain precautions being taken during the removal of such pillars, but his views were not accepted at the time.

On the motion of the Chairman a vote of thanks was accorded to Mr. Redmayne, and discussion of the paper was adjourned.

Fig. 1.

Map shewing the Coal Deposits of the Colony of Natal.

Scale of English Miles

NOTE The figures on the Map shew the heights above the Sea Level in feet

Railways opened
Mountain Ranges
Rivers

SOUTH AFRICAN REPUBLIC
ORANGE RIVER FREE STATE
BASUTO LAND
GRIQUA LAND EAST
PONDOS
ALFRED
ZULU COUNTRY
PIETERMARITZBURG
DURBAN
INDIAN OCEAN
Newcastle Division
Klip River Division
Umsinga Division
Umvoti Division
Umgeni Division
Lions River Division
Upper Umkomas Division
Ixopo Division
Alexandra Division
Tugela Division
Victoria Division

Newcastle 3854 Ft 1168 M
Ingogo 2810 Ft 856 M
Dundee 4043 Ft
Ladysmith 3284 Ft 1000 M
Elands Laagte 5612 Ft 2054 M
Estcourt
Greytown
Weenen
York
Ingeleton
Howick
Pietermaritzburg 2218 Ft 676 M
Verulam
Durban 57 Ft
PORT NATAL
Umgeni R
Scottsburgh
Umzinto
Alexandra
Richmond
Champagne Castle 10,857 Ft
Giants Castle 9657 Ft
Tugela R
Buffalo R
Mooi R
Umkomas R
Umzimkulu R
Umtamvuna R
Drakensberg Range
Kahlamba
Van Reenen's Pass
Orange Free State Boundary
Transvaal Boundary
Natal Coal
Coal

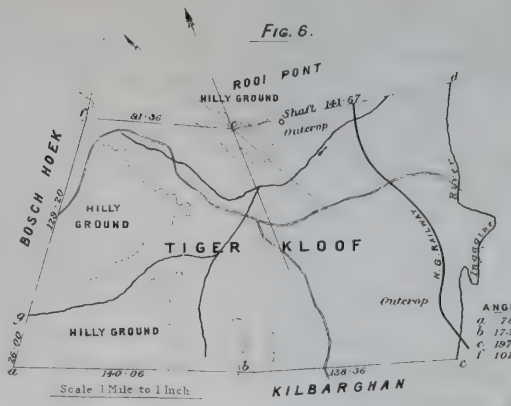
Fig. 2.

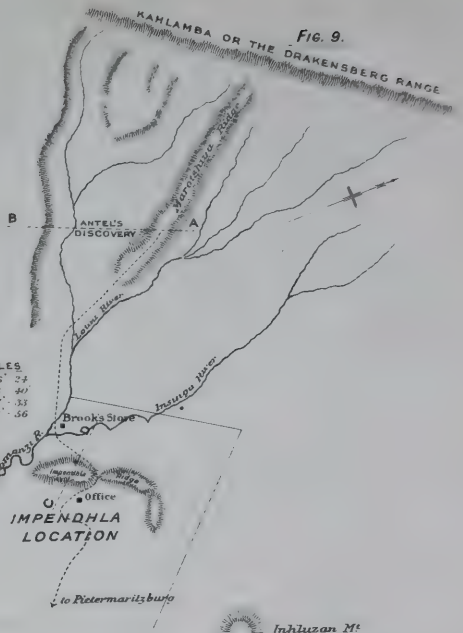
Section along the Main Road from the Drakensberg to Port Natal, Plan to 6 Miles = 1 Inch. Vertical Scale 1 Inch = 1000 Feet.

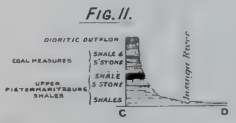
Longitude East 30° of Greenwich

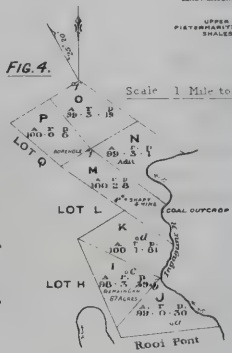
Vol. XLII Plate VII

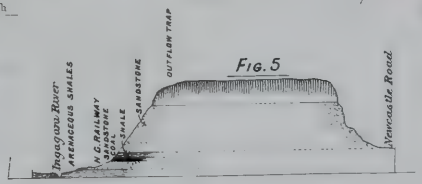
To illustrate Mr R.A.S. Redmayne's Paper on "The Geology and Coal Deposits of Natal."

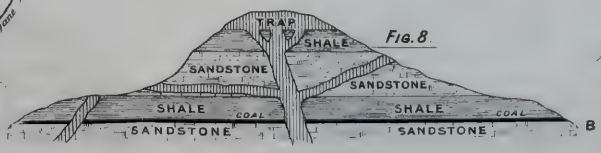


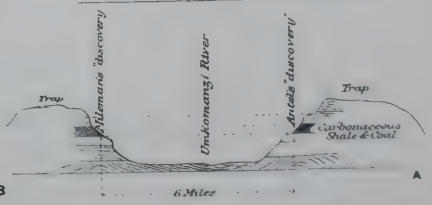












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NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

GENERAL MEETING,

HELD IN THE WOOD MEMORIAL HALL, NEWCASTLE-UPON-TYNE,

JUNE 10TH, 1893.

MR. J. B. SIMPSON, PRESIDENT, IN THE CHAIR.

The SECRETARY read the minutes of the last General Meeting, and reported the proceedings of the Council at their meetings of May 27th and of that day.

The SECRETARY read the Balloting List for the election of officers for the year 1893-94.

The following gentlemen were elected, having been previously nominated :—

MEMBERS—

Mr. H. M. BECHER, Mining Engineer, 9, D'Almeida Street, Singapore.

Mr. J. C. S. BEYNON, Civil and Mining Engineer, P.O. Box 1364, Johannesburg, Transvaal.

Mr. GEORGE DAVEY, Mining Engineer and Metallurgist, Ferro-carril Michoacan y Pacifico, Maravatio, Mexico.

Mr. STANLEY H. FORD, Mine Agent, Stolzenfels, Great Namaqualand, South Africa.

Mr. THOMAS GILCHRIST, Mining Engineer, Manor House, Penshaw.

Mr. GEORGE LINDAY, Colliery Manager, Blackett Colliery, Haltwhistle, Northumberland.

Mr. WILLIAM ERNEST LISHMAN, M.A., Viewer, Bunker Hill, Fence Houses.

Mr. J. S. QUIRK, Metallurgist, St. Helens Lead Smelting Works, St. Helens, Lancashire.

Mr. F. GEORGE SHAW, Mining Engineer, London.

Mr. EDMUND SPARGO, Mining Engineer, 3, Cable Street, Liverpool.

Mr. CHARLES ROBERT WESTERN, Engineer, Broadway Chambers, Westminster, London, S.W.

Mr. JOSEPH HENRY WOOLCOCK, Civil and Mining Engineer, 49, Lowther Street, Whitehaven.

ASSOCIATE MEMBERS—

Mr. G. A. FERGUSON, Editor of the *Mining Journal*, 18, Finch Lane, London, E.C.

Mr. HENRY M. JAMES, General Manager, Colliery Office, Whitehaven.

Mr. HARRY PAGE WOODWARD, Government Geologist, Perth, Western Australia.

ASSOCIATES—

Mr. JAMES BAINBRIDGE, Under Manager, North Walbottle Colliery, Newcastle-upon-Tyne.

Mr. ROBERT EMERSON, Engineer, Tudhoe Colliery, Spennymoor.

Mr. ALEXANDER A. JAMES, Under Manager, Croxdale, near Durham.

Mr. JOHN W. THOMPSON, Colliery Surveyor, Backworth Colliery, Newcastle-upon-Tyne.

The following gentlemen were nominated for election :—

HONORARY MEMBER—

Professor JOHN HERMAN MERIVALE, Professor of Mining, Durham College of Science, Newcastle-upon-Tyne.

MEMBERS—

Mr. ARCHIBALD THOMAS BROWN, Mining Engineer, 372, Flinders Lane, Melbourne.

Mr. WESTGARTH FORSTER BROWN, Mining Engineer, Alston House, Parade, Cardiff.

Mr. RICHARD ECK, Mining Engineer, Beaconsfield, South Africa.

Mr. JOSEPH GOULDIE, Mining Engineer, Kimberley, South Africa.

Mr. JOSEPH HARGREAVES, Colliery Manager, Gwaun Cae Gurwen Collieries, Brynamman, R.S.O.

Mr. JOHN HOLT, Jun., Civil and Mining Engineer, The Hollies, Heywood, Lancashire.

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Mr. EDWARD HALSE read the following paper on "The Gold-bearing Veins of the Organos District, Tolima, U.S. Colombia."

THE GOLD-BEARING VEINS OF THE ORGANOS DISTRICT, TOLIMA, U.S. COLOMBIA.

By EDWARD HALSE.

Introduction.

Organos is a small Indian village lying in a valley near the pass of Chiflón,* one long day's mule-ride south-west of Aipe, and about the same distance north-west of Neyva—both towns on the upper Magdalena river. Savanilla is the port for this river, which runs between the eastern and central cordilleras of the Andes northward to the Caribbean Sea. The Magdalena river is navigable for light-draught steamers as far as Honda (650 feet above the sea), where the first rapids are met with, a distance of about 550 miles from the coast. Above Honda smaller steamers ascend to Girardot, the port for the capital, Santa Fé de Bogotá (8,300 feet) on the eastern side, and for Ibagué (4,300 feet) on the western side; and, during the wet season, to Neyva (1,200 feet), about 700 miles in the interior.

The Andes in Colombia are divided into three ranges or cordilleras. These are spread out considerably northward, but run close together to the south. The central cordillera is *par excellence*, the metalliferous one. The mines of the Organos district are situated on its eastern flank. To reach these from the Magdalena river one has to cross the Chiflón Pass over a singularly bold and serrated range running north-and-south to the west of Aipe and Neyva. The range, rising from 7,000 to 8,000 feet above sea-level, is composed of highly-inclined beds of a hard red conglomerate alternating with sandstone, whose trend is parallel to the axis of the range, and whose dip is eastward. The formation is said to be of Triassic age.

Owing to the hardness of the rocks and the steepness of the mountain-sides, the road over the Chiflón Pass is one of the worst in Colombia, bad as

* A ***chiflón*** is (1) stagnant water into which a stream falls with noise; (2) a wooden pipe or flume; (3) a mining term, *trabajar chiflón*, is work making way in length and depth.

the roads of that country generally are; and mules laden with cargoes not infrequently slip down the western slope, and are lost in the ravines below. From the top of the Chiflón Pass a splendid view of the surrounding country is obtained. Eastward one looks down the forest-clad side of the mountain, dotted below with magnificent royal palms (*palmas reales*) and other tropical trees, across wide plains down the centre of which the Magdalena river winds, until the eye rests upon the eastern Andes, blue and hazy in the distance. Westward the topography is entirely different. In this direction one looks upon a sea of almost treeless hills, backed by forest-clad mountains towering one above the other, the whole forming part of the eastern flank of the central cordillera. The hills in the foreground, covered only with grass and small shrubs (except in the valleys and gullies, the lower portions of which are more or less densely clothed with vegetation) form steep and narrow ridges, trending north and south, flanked by knife-edged cordons or ribs (*cuchillos*) oblique or perpendicular to them, which in their turn are strongly ribbed in a direction perpendicular to their length. Thus between the ranges are narrow winding valleys or cañadas, down the moderately-sloping bottom of which a river or strong stream of water usually courses, and between the lateral cordons are steep gullies or *quebrados* opening into the valleys on either side, down which torrents of water flow in the wet season.

The formation consists of crystalline eruptive rocks and metamorphosed schists. Of the former granophyre,* felsite, and hornblende-dolerite, in bedded masses and in dykes, predominate. The granophyre and felsite are generally moderately hard, and of a pale green colour. They are in places stained red with oxide of iron, and are frequently altered to a soft white clay resembling kaolin. The dolerite is usually very hard, and of a darker green—in places this rock too is soft and decomposed, but it usually retains its colour. Wherever observed it is strongly charged with iron pyrites. It is chiefly from the rapid erosion of some of the beds that the hills derive their peculiar topography.

The metamorphic rocks, usually consisting of chloritic schists and slates, are seen at the base of some of the hills, and form the major portion of others.

The schistose rocks in the State of Tolima have been variously described

* A mixture of orthoclase-felspar and quartz which have simultaneously consolidated. These rocks in the Organos district were formerly called sandstone. Sections cut of typical specimens of the country have been examined and named by a well-kown petrologist. Some exhibit beautiful examples of the structure known as granophyric.

as of Silurian, Cambro-Silurian, and Laurentian age. These and the eruptive rocks overlying them are most probably Palæozoic, but so far in sufficient data have been collected to determine their exact age. It appears that when the central cordillera was upheaved at the close of the Triassic or commencement of the Jurassic epoch, these beds, breaking through the Triassic sandstones overlying them, were forced up to the surface, and were considerably contorted during the process. The formation in places exhibits evidence of a good deal of confusion; the strike and dip of the beds vary considerably, and at short intervals; they are also much jointed and fissured, and are moreover permeated by felsitic dykes.

Here and there in the eruptive rocks are lenticular masses of schistose rocks, and the former appear to merge imperceptibly into the latter, so that possibly the rocks called eruptive are the result of extreme metamorphism of sedimentary rocks brought about by long-continued pressure, heat, and aqueous action.

A number of veins have been found traversing the rocks in various directions--the prevalent one being nearly east and west—and containing quartz with visible gold associated with manganese and oxide of iron near the surface, and with iron pyrites, galena, etc., below. The gold above the water-level is extremely patchy, and visible gold is by no means the sign of a rich vein, for it is seen here and there in veins whose average only runs a few pennyweights to the ton. Some of the veins run into and continue through the schistose rocks, and appear to be of more stable nature therein. Some veins are very rich, especially at and near the surface, carrying gossan with from one to several ounces of gold to the ton for a certain distance, which seldom amounts to many yards either in length or depth, and then a fresh vein has to be sought elsewhere. Mining in this will-o'-the-wisp style was carried on for some years, and has only recently given place to something approaching to system; but as the water-level has not yet been reached, the problem has still to be solved whether the veins are sufficiently auriferous to be worked profitably for any long period of time.

So far as the writer is aware, a description of this district has never yet been published. Señor Vicente Restrepo, in his very interesting work* on the Colombian mines, mentions only the fact that numerous gold and silver veins occur here, and that in Constancia one only was then being worked containing rich ore, but scarce.

* *Estudio sobre las minas de oro y plata de Colombia*, second edition, 1888. page 99. Translations of this work have appeared in English and French.

The veins may be classified as follows :—

1. Bedded or segregated veins (*Lagergänge*) of rich auriferous quartz or gossan.
2. (*a*) Veins of rich auriferous gossan } Usually of shallow depth and uncertain length : they may be termed superficial fissure-veins.
   (*b*) Veins of rich auriferous quartz }
3. Auriferous flucany joints.
4. Quartz fissure-veins carrying free-gold near the surface, and auriferous pyrites below.

Classes 1, 2, and 3 generally occur in the eruptive rocks only, while class 4 occurs both in the eruptive and sedimentary rocks beneath or adjacent to them.

1.—Auriferous Bedded Veins or Seams.

Typical veins of this class are seen at the Chorro working of the Constancia mine, where a big open cut has exposed several of them, and from which a fair quantity of gold has been raised. Fig. 1, Plate IX., is a section of one of these auriferous quartz-seams which, at the point examined, is from 6 to 18 inches thick, and suddenly changes in dip from almost horizontal to nearly vertical. It appears to be in reality a somewhat complex case of a saddle-reef. The country here is decomposed granophyre stained with iron and manganese, some of whose layers are seen to end abruptly against one wing or leg of the saddle. Such contortions are local in character, and the veins much more frequently have the appearance of Figs. 2 and 3, Plate IX. In the former case the vein is about 1 foot wide, consisting of quartz in two layers lying on a thin stratum of rock veined with quartz, oxide of manganese, and ochre. The true floor is altered, ochre-stained granophyre, while the roof is still softer, being composed of ochreous and manganiferous clay.

The quartz-layers are apt to merge suddenly on either side into rock, leading to the inference that they were formed by replacement. The quartz contains iron pyrites in its cavities, and sometimes clay and gold, as well as blende and galena stained with oxide of iron. The gold in these seams is generally visible, and sometimes coarse. It is irregularly distributed, and some very rich specimens can be found.

The bedded veins run from nearly east and west* to a little west of north, and dip about 60 degs. south or west respectively (variation 43 degs.

* All bearings magnetic, 1888.

to 70 degs.). The direction of one bed here is north 35 degs. east to south 35 degs. west (dip south-east 43 degs.), which is nearly normal as regards both strike and dip.

A slickenside joint clearly marked, being stained with black oxide of manganese, is sometimes visible in the clay forming the roof (Fig. 3, Plate IX.). It was noted that the lines are not parallel with the dip, but make an angle of about 10 degs. with it, and also that in places there is a want of parallelism among the lines themselves. The seams are crossed perpendicularly by some joints also showing slickensides. As will be seen from the section (Fig. 3) the country adjacent to or separating the quartz-layers is decomposed to a clay.

In places the rock shows evidence of a schistose structure, but this matter will be referred to again when treating of veins of class 2.

At the Silencio mine at least two bedded auriferous quartz-veins are seen to crop out on the face of a very steep hill. The lower one is the richer and can be traced for about 100 yards along the surface. The outcrops as exposed on the hillside form an angle of about 25 degs. with the horizon. The chief bedded vein runs about north $8\frac{1}{2}$ degs. west to south $8\frac{1}{2}$ degs. east, and dips west 30 degs. to $48\frac{1}{2}$ degs., or against the hill. The average width of the vein is 2 feet, but the rich auriferous quartz-leader is seldom more than 6 inches thick, the rest of the vein being composed of smaller layers of quartz divided by country. The quartz varies in structure from solid to sugary, and is spotted with iron pyrites, manganese, oxide of iron, clay, etc. Visible gold is of common occurrence in certain parts of the outcrop. In a piece of highly auriferous quartz a single crystal of hæmatite was found. The vein was sunk upon in one place—the dip changed from 30 degs. at the surface to $48\frac{1}{2}$ degs. below—the quartz quickly wedged out a few yards from the surface and with it the gold, giving place to a clay or flucan. The latter was followed down to a depth of about 100 feet on the dip, when the sinking was stopped, as no further ore had been met with. The veins vary in richness from about 3 dwts. to 3 ounces of gold to the ton, and are usually rich or poor on the same horizon.

At the Socorro mine a bedded vein has been partially worked, which in one place has the section shown in Fig. 4, Plate IX. It appears to be a trough or inverted saddle-reef. The quartzose seam merges in places into ferruginous rock, and some layers of the country with thin seams of quartz end abruptly against one wing or leg of the trough. In neither of the peculiar contortions figured is there evidence of a saddle or trough-joint.

2.—(*a*) Veins of Auriferous Gossan.

At the Te Encontré* workings of the Constancia mine there are several examples of this class of vein in quartz-felsite. They are remarkable for their very singular appearance (Figs. 5 to 10, Plate IX.). They trend usually a few degrees north of east, and dip south, and, as a rule, are auriferous to no great extent either horizontally or vertically. They appear to be especially auriferous when crossing certain manganiferous, ferruginous, and ochrey layers of the country (Figs. 5 and 6), which strike a little west of north, or perpendicular to the veins, and against certain wedges or horses of schistose rock found in the felsite (Figs. 7 and 10).

The ore is generally gossan or earthy brown oxide of iron, which frequently contains visible gold. The centre of the vein is sometimes composed of crystalline quartz. The precious metal is also found in the manganiferous and ochrey layers on the sides of the vein, and even for some little distance therefrom. These veins are extremely irregular, having no defined walls, although generally a southerly dip is roughly traceable.

Immediately above the section shown in Fig. 7 is a layer of soft manganiferous rock 1 foot thick. In Fig. 8 the vein consists of a rich auriferous band of gossan from 12 to 15 inches thick. It is nearly vertical, the south wall being defined by a thin seam of clay or flucan, while no joint occurs on the north side. The vein in all has three nearly vertical seams of clay crossing others dipping flatly to the south; the latter appear to have slipped down a little on the south wall of the vein.

The felsite country rock often has a brownish mottled appearance. Under the microscope it is seen to be traversed by minute stringers of quartz.

At the Socorro mine some thin but extremely rich veins of this class have been met with, consisting of a few inches of light brown gossan showing visible gold in the broken surfaces and assaying 50 ounces of gold to the ton and upwards.

At the Silencio mine several veins of the same class occur in granophyre. One trends on an average east $13\frac{1}{2}$ degs. south to west $13\frac{1}{2}$ degs. north, and dips south 56 degs. to 68 degs. At the surface it is about 1 foot thick, with from 2 to 4 inches of brown gossan on the hanging-wall, the rest of the vein being soft altered granophyre with hard lumps of quartz, country, and oxide of iron here and there. Ten feet down the vein shows iron pyrites, with oxide of iron, and quartz in lumps. It was

* Literally, "I have found thee!" the exclamation of the miner who discovered the first rich vein in this district.

proved by a cross-cut driven in 20 feet vertically below the outcrop. When cut it was found to be disordered by a slide, and consisted of gossan and quartz with well-defined walls having a regular dip. The mineral showed visible gold, and gave an excellent colour in the batea. Westward it was split into two branches by a horse of rock—the branches together made 8 inches of rich gossan (Fig. 9). In this instance the country was intersected by joints more or less parallel to the vein, and dipping in the same direction at an angle of 45 degs. The ore in one spot made against a bed of dark stained rock (altered schist?) as in the Constancia mine (see Fig. 10, and compare Fig. 7). Westward the vein was cut out 24 feet from the cross-cut; eastward, although the joint was straight and well-defined, the vein 20 feet from the cross-cut consisted only of 4 inches of auriferous gossan. A second cross-cut was run in from the surface in order to ascertain if the vein went down. But nothing was cut, not even a pronounced joint, and yet the vein was traceable on the surface for about 100 feet in length.

2.—(*b*) Veins of Auriferous Quartz.

At the same mine, a vein believed to belong to this class intersects the bedded veins of gold-bearing quartz already described as running north 8½ degs. west, and dipping west. The vein trends north-east to south-west, dips south-east from 30 degs. to 50 degs., and consists of what is known locally as *formacion*, or country altered to a clay (the *cascajo* of Venezuelan, the *mullock* of Australian, and the *bluff* of American miners). At the junction it was 1 foot wide, and when driven on into the hill in a south-westerly direction the vein was double in size, and had much improved in value. Fig. 11, Plate IX., shows the appearance of the vein some little way in. The formacion here consists of decomposed granophyre, mottled red, yellow, white, and brown, and containing native gold in crystals. The quartz was fairly spangled with gold, almost every stone broken showing the precious metal; but very loose, and in exceedingly thin plates and fine grains. The gold was easily shaken out of the specimens, and appeared to have been deposited quite recently. A sample taken, not showing visible gold, gave 1 oz. 8 dwts. of gold to the ton. The decomposed rock in the vein gave a good colour in the batea.

Farther in the vein showed a tendency to split up (Fig. 12, Plate IX.), and was less rich. However, the vein has since been driven on for a length of 65 feet, the formacion carrying gold, and increasing in width with stringers of ore. A rise was put up 43 feet to the surface; for 28 feet

the vein was very rich in gold, and from 2½ to 4½ feet thick, the last 15 feet consisting of formacion carrying gold and intersected with stringers of quartz to within a few feet of the surface.

The vein was sunk on 25 feet, the lode, including formacion, was from 3 to 3½ feet wide with solid quartz from 1 to 2 feet wide. For a time the lode improved in depth, every stone broken showing visible gold. Nevertheless the writer believes that eventually the vein gave out both in length and depth.

This vein, consisting as it does of quartz and altered rock (formacion), and containing little or no gossan, seems to form a class of its own.

3.—Auriferous Flucany Joints.

The only auriferous joints observed by the writer in this district are a few at the Socorro mine. These run east 19 degs. north to west 19 degs. south, and are vertical, or dip sharply to the south. The joints bear clay and native gold, but are auriferous to only a very limited extent. They possess features of no particular interest.

4.—Auriferous Quartz Fissure-veins.

These veins are found traversing the eruptive and metamorphic rocks indiscriminately. Two veins of this class cross the bedded veins of Chorro, one trends north-west with a high dip, is 27 inches thick, and consists of quartz, and red, yellowish, and black-stained rock; the other vein strikes east 22½ degs. south to west 22½ degs. north, and in one portion is filled with clay and oxide of iron rich in gold, while in another it is 2 feet wide, and is filled with quartz, clay, and iron pyrites.

At the Socorro mine one vein which has been proved near the surface for about 200 feet in length occurs in more or less altered granophyre. The average run of the lode is east 26 degs. north to west 26 degs. south, the average dip south 72 degs., and the average width 3 feet. The strike varies from east 16¼ degs. north to east 35½ degs. north, the dip from 62 degs. to 82 degs., and the thickness from 4 inches to 6 feet. In the granophyre the contents are chiefly auriferous oxide of iron and quartz in leaders, separated by more or less altered rock (Fig. 13, Plate IX.).

About 100 feet below the outcrop the lode passes into hard hornblende-dolerite—possibly an altered schist; the dip changes from 72 degs. to 82 degs., and the lode is characterized by bands of iron pyrites, galena, quartz, and hard country rock—the latter either dead, or with some calcite, quartz, and iron pyrites. Each wall is marked by a well-defined ferruginous joint, but without any clay selvage. Fig. 14, Plate IX., shows

the structure. A joint or bedding-plane striking north-west, dipping south-west, and crossing the lode near the hanging-wall, is probably the feeder of the lode at this point. The country next to this wall is sprinkled with iron pyrites, and contains calcite in large crystals.

The mineral from this portion of the lode, hand-picked, yielded 6 ounces of gold to the ton and upwards, and paid to export to Swansea.

The trend of the lode in the moderately hard rock, and where rich, is east 17 degs. north—this is the useful bearing of the lode—but in very hard rock met with farther west the trend becomes more northerly or east 35 degs. north (angle of horizontal bending = $19\frac{1}{4}$ degs.), the dip changes to north 80 degs. (angle of vertical bending = 18 degs.), and the lode is impoverished. Furthermore, a cross-joint is met with here, which has the effect of cutting off the mineral contents almost entirely. Harder rock is met with in the stopes also in this direction, and the lode is poorer, but a few hundred feet farther west some rich quartz from 1 to $2\frac{3}{4}$ feet in width is found in the lode near the surface (rock : decomposed granophyre), where the trend and dip correspond to that of the richer portion of the lode in the dolerite.

A cross-cut driven north-west through the country in this part of the mine discloses purple beds of quartzose schists trending north-west to south-east and dipping south-west, followed by a dyke of hard dolerite dipping northward, which is succeeded by altered granophyre. Here and there also in the schists occur patches of the latter rock.

At Te Encontré, a lode has been proved for about 600 feet in length and 182 feet in depth in quartz-felsite. The average run of the lode is east 9 degs. north to west 9 degs. south, and the average dip south 63 degs. The angle of total divergence in strike is $31\frac{1}{2}$ degs , and in dip 49 degs The average width was 5 feet, varying, however, from 6 inches up to 6 feet. In the No. 1 and No. 2 levels, the rock is quartz-felsite, often much altered, and the vein consists of ferruginous quartz with free-gold. In the No. 3 level, however, the rock is dolerite, which becomes very soft in the No. 4 or bottom level.

In No. 3, where the rock is moderately hard dolerite, the lode is about 6 feet thick, half of this being well impregnated with galena and iron pyrites, the latter being scattered all through the lode. The galena is argentiferous and carries some gold, but the iron pyrites is much more auriferous. The effect of a flucan crossing the lode in one of the stopes is to flatten it considerably (Fig. 15, Plate X.). At *a*, the vein was continued inside the clay flucan itself, as an irregular mass of auriferous

and ferruginous quartz. In the No. 4 level, or 182 feet from the surface (measured vertically), a cross-course has thrown the lode 110 feet to the north. On the west of the cross-course, the lode is $4\frac{1}{2}$ feet wide in a soft green rock with joints having polished surfaces (altered dolerite) sparsely sprinkled with iron pyrites. The dip is south 79 degs. The lode to the east of the cross-course is quartzose, from 12 inches to 2 feet thick, and poor in gold.

About 12 feet south of the western portion of the lode in No. 3, a vein of quartz 2 feet wide and dipping south is seen on the walls of the cross-course, and 24 feet farther south is another vein of quartz, somewhat auriferous. As these veins pass through the cross-course they would appear to be of more recent date than the latter.

Hence in this mine we appear to have evidences of :—

1. An older east and west fissure.
2. A newer north and south cross-course.
3. Newer east and west fissures.

About 200 yards east of this, a parallel vein was discovered of 2 feet of white sugary and friable quartz, showing here and there minute specks of native gold. Notwithstanding the favourable appearance of the quartz, the yield was only a few pennyweights to the ton. It was considered likely that most of the gold had quite recently been leached out of the vein.

At Silencio, a vein of this class, traceable for 600 feet along the surface, has been worked 400 feet in length at a depth of about 100 feet vertical from the surface. The average strike, dip, and width are east 29 degs. south to west 29 degs. north, south-west 62·5 degs., and 2 feet respectively. The rock on the 100 feet level is in the main hard granophyre. The lode was widest (4 to 6 feet) and richest when the strike was north $42\frac{1}{2}$ degs. west or north-west nearly, but to the north of this point a branch falls into the lode on the hanging wall, and near the centre of the run of ground, a non-auriferous quartz-vein trending nearly east and west and dipping south, crosses it obliquely. The lode was formed of one or more layers of quartz sprinkled with iron pyrites and galena, and showing here and there crystals of blende. The quartz was generally of somewhat transparent crystalline structure, occasionally sugary, and frequently stained red in the joints. It was colourless, or had a brownish tinge. Sometimes the quartz was light bluish. Here and there it contained visible gold, but the latter was no real indication of richness. The gold contents varied from a few grains to several ounces to the ton. A milling test of several hundred tons yielded about 4 dwts. to the ton,

and as 1 dwt. was found in the tailings, the average gold contents were only $\frac{1}{4}$ ounce to the ton.

Just beyond the point where the lode was widest it split up into two branches: that on the southern side consisted only of a few inches of unproductive quartz, while the northern branch consisted of about 6 inches of auriferous and pyritic quartz. The hanging-wall appeared to be a schistose rock, the footwall being brown granophyre. In some points the schistose rock was on the footwall, but it was more generally on the hanging wall. The leader sometimes lay on the hanging wall, but it was more often a little distance from it, and sometimes having a much flatter dip, it gave the lode a wedge-shaped appearance as seen in section. Figs. 16 to 20, Plate X., represent the appearances assumed by the lode in this part of its course. By comparing Figs. 16 and 17 (the latter is from a point about 1 foot north of the former) it will be seen that a band of rock highly charged with iron pyrites has become a band of quartz charged with the same mineral; this appears to be an instance of local replacement.

It has been observed that in certain districts galena is a good indication for gold.* In order to ascertain whether the galena contained the precious metal, the writer made (1) an assay of the quartz well spotted with this mineral, the result was 9 dwts. of gold to the ton; (2) an assay of galena carefully picked out from the quartz resulted in 45 ounces of silver and 2 dwts. of gold to the ton. As the concentrated sulphides in the vein yielded nearly 15 ounces of gold to the ton, the result was arrived at that the major portion of the gold is present in the pyrites, while some is undoubtedly free at and near the surface.

A long cross-cut was driven in to strike this lode at a depth of about 200 feet from the surface.

In the cross-cut, three hard bands of hornblende-olivine-dolerite were cut bearing iron pyrites alternating with granophyre varying from moderately hard to soft. The rocks appeared to be in beds from 3 to 20 inches thick, trending east 10 degs. south and dipping south 36 degs. to 53 degs. A few joints cut these beds striking north $22\frac{1}{2}$ degs. west to north-west, and dipping south 54 degs. Others ran east and west to east $22\frac{1}{2}$ degs. south, dipping north 51 degs. to 62 degs.

* "It is worthy of note that in this mine (Latham and Watson's quartz mine, on Hustler's Reef, Sandhurst) the gold is aggregated in cavities with the softer sulphides, such as galena, and is rather sparingly diffused in the laminæ which enclose iron pyrites."—R. Brough Smyth, *Gold Fields and Mineral Districts of Victoria.*, 1869, page 329; see also footnote on same page. Galena is often associated with free-gold in the New Morgan gold-mine of Wales.

According to recently published reports, highly contorted ground has been met with just where the lode was expected to be cut, consisting of 9 feet of blue auriferous quartz and 15 feet of limestone-deposits (*sic*), the latter also carrying some gold.

In the north-western portion of the mine, the rock has a brownish mottled appearance; examined through a microscope it is seen to be granophyre spotted with chlorite (probably after biotite). In this section the veins bear very little gold.

In the same mine, a pyritous quartz-vein coursing through chloritic-schist was proved for about 135 feet in length. It trended on an average east 34 degs. south to west 34 degs. north. The filling was 18 inches of white quartz, well charged with iron pyrites, but the average assay value was only about 10 dwts. of gold to the ton.

On the opposite side of the river Chiquilá, several veins are seen trending through clay-slate rocks, some of which have been worked to a certain extent. In the bottom of the Arroyo del Muerto, separating the La Reina from the La Union claims, two veins appear to form a junction, V-shaped in plan. The easternmost of these consists of 8 inches of quartz sprinkled with sulphides. In a level 30 feet above, both the quartz and the country are loose in texture. The quartz is white, and in one leader 8 inches thick below, 7 inches in the middle, and 3 inches at the top of the level. The vein trends north-west and dips south; outside the level, the vein is split up and disordered. Higher up the hill eastward are two outcrops which appear to be continuations of the western branch of the same vein. The first and lower one is 9 inches thick of brown quartz dipping south 24 degs.; the second and higher one of similar quartz 6 inches wide and dips south 39 degs. A sample from these out-crops gave 9 dwts. of gold to the ton.

The westernmost vein runs east 10 degs. south to west 10 degs. north, and dips south 50 degs. on an average. The vein is traceable along the slope of the hill for a length of 300 feet, corresponding to a height of 160 feet vertical. At the top working, the vein consists of 6 inches of solid quartz dipping south 73 degs.; 50 feet below this point, the dip changes to south 47 degs., and the vein averages 10 inches. The bedding-planes trend a little north of east, and dip south 62 degs., so that the vein forms an acute angle with them. 25 feet farther down, the vein consists of white quartz sharply curved in a double-cleaved or jointed rock (Fig. 21, Plate X.). The average dip is south 30 degs. In the lowest working (100 feet below the first) the dip is south 51 degs., and the thickness is 2 feet of white quartz strongly cleaved and jointed. Samples from the

surface-workings averaged 5 dwts. of gold to the ton. Old workings exist under this vein which were inaccessible at the time of the writer's visit.

A tunnel, 600 feet above the river Chiquilá, and a little west of these workings, has cut two veins. The first was cut 50 feet from the mouth, it trends north 22½ degs. west, and dips west about 45 degs. For some distance the lode was auriferous, consisting of 21 inches of white and comparatively tender quartz sprinkled with iron pyrites. Southward the lode takes a sharp turn east, and here the lode is much flatter, consisting of whitish quartz sprinkled with a little iron pyrites. The rock is bluish clay-slate. The vein averages 5 to 6 dwts. of gold to the ton; northward it is soft and worthless. About 20 yards farther west another vein or branch runs north-east and south-west, and dips south 45 degs., consisting of one foot of white quartz of very favourable appearance sprinkled with iron pyrites. It averages 10 dwts. (extremes, 5 dwts. and 3½ ounces) of gold to the ton. It must form a junction with the first vein to the south, and is said to have done so in one of the stopes above, where rich mineral was found.

Near the eastern boundary of the La Virginia claim, a quartzose vein, 16 inches thick, is seen crossing the Arroyo north 35 degs. west to south 35 degs. east in clay-slate country. The vein has been proved superficially along the slope of the hill for a length of 350 feet, corresponding to a vertical height of 125 feet. At the top working, the vein is seen to be a red and yellow-stained dyke of felsite with a little gossan and quartz 3 feet thick in hard clay-slate rock (Fig. 22, Plate X.). 30 feet below this, the vein courses north-west, and dips south 72½ degs. in a similar hard country. The vein-formation is 3½ feet thick, the auriferous quartz consisting of a single layer lying near the footwall (Fig. 23, Plate X.). Divisional planes are traceable here trending east and west, and dipping south 75 degs. 70 feet below the last working, the vein consists of only 2 inches of quartz spotted with iron pyrites. The beds of the country here run east 35 degs. south to west 35 degs. north, and dip to south 51 degs. An average sample of the whole vein gave only a few pennyweights of gold to the ton. The country here is clearly uncongenial, and the vein is pinched below.

At the California mine, an auriferous quartz-vein is traceable for about 350 feet on the slope of a hill, corresponding to a vertical height of 150 feet. The lode strikes east 20 degs. south to west 20 degs. north, and dips from north 68½ degs. to south 58 degs. At the top working, the vein is 6 feet wide, consisting of 2½ feet of hard quartzose gossan with

iron pyrites on the hanging side, the rest being gossany clay (Fig. 24, Plate X.). The pyritous quartz assayed 2 ounces 13 dwts. 8 grains of gold to the ton, while the gossany clay only yielded about $\frac{1}{2}$ ounce. In the croppings below the vein is very thin, consisting of quartz, gossan, and iron pyrites from 2 to 9 inches thick, the assays varying from $1\frac{1}{4}$ to 9 dwts. In one place the quartz is white and non-mineralized.

At El Dorado, about 125 feet above the stream, a vein crops out trending north $32\frac{3}{4}$ degs. west, and dipping south-west 63 degs. It consists of ribs of quartz separated by layers of rock, and with gossan on the footwall. Samples from here have yielded from 9 dwts. to upwards of 5 ounces of gold to the ton. About 200 feet above this, another vein crops out bearing east 11 degs. south to west 11 degs. north, and dipping south $42\frac{1}{4}$ degs., from 12 to 18 inches thick of solid quartz sprinkled with iron pyrites. Assays from here gave only from 4 to 8 dwts.

At Concepcion, a vein trending east $21\frac{1}{2}$ degs. south and dipping south 44 degs., is seen cropping out in a *quebrado;* it consists of 20 inches of hard quartz, with iron pyrites and galena. The hanging-wall appears to be hard granophyre, and the footwall hard hornblende-dolerite. A few feet above, the vein thins away to 9 inches, while about 18 feet higher up, it appears only as 2 inches of soft green clay charged with two iron pyrites.

The prevalent direction of the veins in this district appears to be a few degrees from east and west with a southerly dip. Several veins trend in a north-westerly direction, dipping south-west. A north-easterly direction appears to be exceptional, as does also a northerly dip (Fig. 25, Plate X.).

Leaving the segregated veins out of account, which usually follow the bedding-planes, no well-defined system of fissure-lodes has as yet been discovered in this district.

The veins run in various directions, and individually many of them are very variable in strike. The beds of the country appear to vary considerably in strike also. At the Chorro workings, some of the beds appear to run north 35 degs. east and to dip south 43 degs., while others trend mostly a few degs. north of east and dip south 62 degs.; at La Reyna, about 10 miles farther south, some beds are seen coursing and dipping the same as the latter, but in the same mine other beds strike east 35 degs. south and dip south 51 degs. At Silencio, on the opposite side of the river, some beds appear to run east 10 degs. south and to dip south 36 degs. to 53 degs.; while at Socorro some beds trend north-west and dip south-west. The rocks are frequently so altered as to have their bedding-planes

obliterated—the complication is increased when, as is often the case, the rocks exhibit divisional planes running in various directions.

The fissure veins are evidently of later date than the schistose and eruptive rocks, as they pass indiscriminately through both.

It is extremely probable that all the veins have been filled by lateral secretion, *i.e.*, the mineral contents, including the precious metals, have come in from the adjacent rocks. Certain bands of the metamorphic rocks as well as of the eruptive dykes are strongly charged with iron pyrites, and this mineral may be looked upon as the main source of the gold in the veins. At and near the surface, the well-known decompositions have taken place, resulting in the production of earthy brown oxide of iron (gossan) and the setting free of the gold.

The association of manganese with gold has frequently been noticed. In this district the gossany veins are often characterized by the presence of this ore, and, in certain instances, they are specially auriferous when crossing bands of oxide of iron, manganese, and clay.

It should be noted that in some countries—Venezuela for example—manganese is looked upon as a sign of poverty.*

In some gold veins the presence of galena is regarded as a favourable indication. This appears to be the case in this district, although the galena itself carries little gold.

Gossan containing visible gold is usually regarded as a favourable feature for the precious metal occurring in depth, but, speaking generally, this is certainly not the case in this district. The gossany veins die out rapidly in depth, while the quartz fissure-veins seldom carry gossan at the surface.

Some of the quartz fissure-veins appear to widen while others seem to pinch out in depth—but the latter may only be a temporary thinning due to hard and uncongenial rock.

Although the quartz fissure-veins sometimes follow the direction of the bedding-planes, they more frequently cut them at an acute angle, obliquely, or at right angles; they sometimes, too, follow the lines of contact between the sedimentary and eruptive rocks, and sometimes follow the dykes for a certain length, as well as the divisional planes.

Mr. John C. F. Randolph† says that no true fissure-veins have yet been discovered in Tolima, and they are all classed by him under the head of bedded-veins. His observations, however, do not appear to have been

* *Les Filons d'Or de la Guyane Française*, L. F. Viala, 1886, page 35.

† "Notes on the Republic of Colombia, S.A.," *Trans. Am. Inst. Min. Eng.*, vol. xviii., pages 205-213.

extended so far south as Organos. Even with regard to other districts this is a sweeping assertion to make, and the writer believes it is not borne out by facts.

Quartz appears to be the only true gangue in these veins, and the metalliferous species are extremely few in number, being confined almost entirely to auriferous iron pyrites, auriferous and argentiferous galena, blende, the oxides of iron and manganese (secondary), and a mineral not yet determined.

In the quartz fissure-veins the band of quartz, which is generally found on or near the hanging-wall, may be looked upon as the filling of the original fissure ; other bands frequently occur adjacent to or near this, filling subsidiary fissures, or are the result of replacement of the country-rock.

The rock on either side of the original fissure is often decomposed and mineralized, and frequently exhibits joints or fissure-planes parallel thereto. These were probably produced contemporaneously, and those that limit the mineral contents may be regarded as the hanging and footwalls of the vein.

The country within the lodes may be looked upon as altered or unaltered rock *in situ;* and the original fissures were probably, in the first instance, mere cracks in the country rock.

In places there is evidence of strong local disturbance in the rocks, due probably to great lateral pressure, which has produced local movements of one bed over another, resulting sometimes even in fracture (Figs. 1 and 4, Plate IX.).

Finally, the behaviour of the quartz fissure-veins may generally be said to be irregular, and its peculiarities are as follows:—

1. The thickness varies considerably, and sometimes at short distances, both horizontally and vertically, from a mere joint-plane to about 6 feet, the latter being rarely exceeded. The average would appear to be about 2 feet.

2. The dip also varies considerably, although its general direction is nearly always south or south-west, according as the vein runs in an east and west or north-westerly and south-easterly direction respectively. The dip is rarely northerly or easterly, and when so the lode is usually marked by poverty.

3. The strike is still more variable, and although some of the veins appear to follow the law of average bearing,* others appear very capricious in this respect.

* Moissenet, *Parties riches des filons.*

4. The walls are sometimes ill-defined, especially in the eruptive rocks. In Figs. 16 to 20 a fairly well-defined hanging-wall is observable, while the quartz leader, which we may regard as a footwall, is often flat and irregular.

Auriferous Impregnations.

In addition to the deposits described in the paper there is another class in the district which may be termed auriferous impregnations. They occur in highly kaolinized granophyre, and run as a rule very irregularly, and appear to possess no walls or joints.

At the Silencio mine, a deposit of this class has been opened near the surface. It seemed to run roughly north-east to south-west, and to dip flatly to the south-east. The rock appeared to have been subjected to the action of thermal waters from which hydrated oxide of iron and native gold had crystallized out. The mineral bore a great resemblance to specimens from the famous Mount Morgan mine of Queensland, but unfortunately it was not found in any quantity. Specimens occur rich in visible gold, but the average does not probably run more than half an ounce to the ton. It is believed that most of the gold occurs in the mineral in a very highly divided condition.

Veins crossing these gold-impregnated beds would probably be found to be highly enriched therein.

The gold in the Organos district contains a good proportion of silver, and is only about 650 fine.

The President asked whether the mines were worked to any great extent, what was the quantity of gold produced, the number of workmen, etc.?

Mr. E. Halse replied that a large quantity of gold was not produced at present in the district described; the value might perhaps be put down roughly at from £10,000 to £15,000 sterling per annum. The mines were being extended, and new veins were being opened out, so that in future the district might become a much more important one than it was at present.

Mr. W. Cochrane asked whether the district was likely to become competitive with other gold-bearing districts of other parts of the world?

Mr. Halse said at present it could not be considered as an important district, but he could not say what it might become with further development.

Correspondence.

Mr. David Burns wrote that the Organos mining-field seemed calculated to intensify the mystery that surrounded the origin of veins, rather than to afford material for a solution of the problem. The features recorded are pretty much what obtain when veins occur in extremely metamorphic rocks, save that the veins appeared to be of a specially weak and fragmentary character, but probably more extended operations and deeper working would show a greater continuity than now appears probable. It is abundantly clear from this paper and from many other observations, that the contents of the bedded veins were to a great extent determined by the surface of the ground, and that consequently they were filled when the surface-configuration was approximately what it is now. This is proved by the contents of the veins altering in depth, and in most cases disappearing altogether. The conversion of pyrites into gossan is easily explained by atmospheric influences, but the displacement of auriferous quartz by flucan, as greater depth is attained, admits of no such solution. Neither does the segregation idea help us at all. Let us suppose that a limited amount of intensely heated and highly fluid mineral substances was escaping from the interior along contact-planes as being the planes of least resistance. So long as these substances remained perfectly fluid, they would allow the rift between the rocks to close behind them, but when they began to cool down and became viscous, they would begin to clog up the vent, those with the lowest melting-point getting nearest the surface. This assumption explained why such veins so frequently disappear in depth and present a distinct succession of mineral in depth. It was inconceivable that pyrites and quartz could traverse the planes of bedding of the country-rock by any other agency than galvanic action. If galvanic action had caused the segregation, would not the metals of the veins be found in continuous ribs, and the metals in the metallic form instead of as ores? Metalliferous veins are generally found intersecting dykes and cross-courses and are thus proved to be of later age, and they may be of a much later period than has hitherto been suspected. Mr. Halse says "the rock on either side of the original fissure is often decomposed and mineralized, and frequently exhibits joints or fissure-planes parallel thereto," but this feature was well explained by the theory of the eruption of vein-stuff from below, and not at all by the author's theory of secretion. Much of the quartz and clay in the veins may be the country rock completely metamorphosed by the hot erupted vein-matter, whereas the rocks just referred to have been less altered. The fissure-veins that appeared to widen in depth are probably

stronger or have had more cover than those that narrow downwards; and those that widen in depth will ultimately narrow if followed far enough; their fluids having been able to expend most of their energies before reaching the surface, while those that are widest at the outcrop have not.

Mr. E. Halse wrote that Mr. David Burns appeared to think that veins were filled by mineral substances ascending them in a more or less molten condition. This seems an extraordinary view to take, and one to which the writer believes the present structure of veins lends no support whatever. On the contrary, all the evidence seems to show that the mineral substances were deposited from chemical solution (presumably from heated waters, but not necessarily so in all cases). The writer has advanced no new theory in his paper, but he has taken the most generally accepted one, that of lateral secretion—first outlined by Delius in 1770, and in recent years propounded with much scientific evidence by Fridolin Sandberger,* and largely supported by such keen observers as Emmons, Curtis, Becker, and others—as explaining best the mode of filling of at least the superficial veins of the district described. No single theory will explain all the phenomena connected with mineral veins, nor can it be expected to do so, for the structure is often extremely complicated, owing probably to the veins having undergone many physical and chemical changes since the first filling.

The President proposed a vote of thanks to the writer of the paper.

Mr. M. Walton Brown seconded the vote of thanks, which was cordially adopted.

The Rev. G. M. Capell read the following paper on the "Manometric Efficiency of Fans":—

* *Untersuchungen über Erzgänge*, 1882, 1885.

MANOMETRIC EFFICIENCY OF FANS.

By the Rev. G. M. CAPELL.

It is now about ten years since the term "manometric efficiency" was brought prominently forward by Mr. Daniel Murgue, and the term is one not generally understood. It refers simply to the relation of the water-gauge to the speed of the blade-tips of a fan, and has no necessary relation to the useful effect of a fan.

Mr. G. Herbst, in speaking of fan tests, specially remarks that high manometric effect does not necessarily imply high useful effect. In fact the evidence of tests by Fan Commissions prove the exact contrary, as will be shown later on. When we consider that the water-gauge and volume represent two sides of a balance, of which the water-gauge represents the longer arm, it seems natural to expect, if the water-gauge be reduced, that the volume of air will be increased, and with it the useful effects at a given speed. With high manometric effect, the fan is not working on a full flow of air, and consequently is wasting power in creating the high vacuum implied in a maximum water-gauge with a minimum quantity of air passing.

In the trials of Guibal fans by the Committee of the North of England Institute of Mining and Mechanical Engineers,* the 50 feet Guibal fan at St. Hilda colliery showed a useful effect of 42·09 per cent., while the manometric effect was 64·1 per cent.; the 40 feet Guibal fan at Pemberton colliery showed a useful effect of 52·95 per cent., with a manometric effect of 52·1 per cent.: the Guibal fan at Cannock Wood colliery showed a useful effect of 47·95 per cent., with a manometric effect of 56 per cent.; and the Waddle fan at Celynen colliery, with a useful effect of 50·5 per cent., gave a manometric effect of 45·7 per cent.

Another consideration comes in next, that is the position of the end of the water-gauge tube. In all the early Guibal fans the water-gauge tube was placed either in the centre of the inlet over the bearing or in the fan case. Now this makes a considerable difference in the calculation of the manometric effect. If another water-gauge were placed on the

* *Trans. N. E. Inst.*, vol. xxx., page 273.

drift, 20 feet from the fan inlet, another and a lower reading would be shown. There is an instructive example in the trials of the 30 feet Guibal fan at Staveley colliery by Mr. Robert Howe,* at 56 revolutions of the fan the water-gauge in the inlet was 2·52 inches, with a manometric effect of 73 per cent.; but the water-gauge in the drift was 2·10 inches, with a manometric effect of 60·3 per cent. There is a difference of about 13 per cent. in the manometric effect, obtained by changing the water-gauge from a position at the fan-inlet to the drift-door, which was close to the fan-inlet. At a distance of 20 feet, the manometric effect would have been much less.

In recent trials of fans by a private commission in Belgium, there is an interesting series of diagrams showing the position of fan-inlets and the water-gauges at various points. At the Rieu-du-Cœur colliery, a Rateau fan has been erected, and this fan is said to show a manometric effect of 80·2 per cent., with a useful effect of from 35 to 47 per cent. The position of the water-gauge tube was peculiar. The conical adjutage, joining the fan to the drift, is of small diameter, not above $4\frac{1}{2}$ feet, and the water-gauge tube is placed in it. With 59,488 cubic feet of air passing, the velocity would be 4,210 feet per minute in the tube, and, by Dr. Hutton's rule, the water-gauge due to that velocity would be 1·70 inches, representing a very large deduction from the 4·29 inches shown, had the gauge been taken on the door of the large drift, from which the fan was drawing. Yet a comparison is made of the manometric effect of that fan, with its water-gauge tube in the position described, with that of a Capell fan at the Prosper colliery, with the water-gauge tube placed about 90 feet from the double inlets of the fan, in a drift of 55 square feet area, the water-gauge showing 6·10 inches.

If manometric effects are to be of any use in comparisons, the conditions of taking them should be settled, as they become misleading when made under wholly different conditions. In the report of the Belgian Commission, only the Guibal fans appear to have been tested under similar manometric conditions.

Another point is worthy of notice, that a very small fan passing a large volume of air has to raise that air to a high velocity; and velocity means work done, and it is fair to measure the work done, remembering that the static water-gauge (*i.e.*, the water-gauge taken out of the air-current in the fan-drift) is the water-gauge which should be effective for the ventilation of a mine. That water-gauge can always be ascertained by taking a reading on the outer door of the fan-drift, the inner door being open. If

* *Trans. Chesterfield Inst.*, vol. i., page 46.

all water-gauges were taken thus, the useful effects would become less, as at present calculated, but the real effective work of fans would become clearer.

The writer has found that the highest useful effects in fans of the double centrifugal type is obtained between 47 per cent. and 54 per cent. of manometric effect. This agrees with the results of the Committee of the North of England Institute on Guibal fans, and brings the useful effects of Guibal fans down to the level of fans placed on Belgian mines by Mr. Guibal. The position of the water-gauge tube in fan tests determines not only the manometric efficiency, but the mechanical efficiency; and the reduction of the mechanical efficiency becomes a serious matter in a case where, with one fan, the water-gauge tube is placed, say, 60 feet away from the inlet of the fan, while with the other fan the water-gauge tube is placed in or close to the inlet. It follows, that in such calculations, some fixed distance from the inlet should be settled for the water-gauge tube end, so as to avoid conclusions which are misleading and inaccurate.

Another point of discussion is whether the water-gauge effect on the mine ventilation is the same with a double-acting as with a single-acting centrifugal fan. Experiments with two fans acting in succession, *i.e.*, one fan discharging into the inlet of another, show that the effect is not the same as with only one fan acting on the mine at a given speed. The writer believes fully in the difference, and this is a point which would at once raise a question as to the real worth of exceptionally high manometric effects.

The manometric rule of Mr. Murgue shows a theoretical gauge double of that given by Mr. Guibal $h = u^2 \div g$; while Mr. Guibal used $h = u^2 \div 2g$.

A curious point comes out in examining these water-gauges and manometric effects. If a fan gives 80 per cent. of manometric effect, then the speed of the air flowing into the vacuum, formed by the revolving blade-tips, is greater than the velocity of the actual circumference of the fan. Take the case of a fan 9·2 feet in diameter, giving 80 per cent. of manometric effect at 228 revolutions and 4·29 inches of water-gauge. The velocity due to a depression of 4·29 inches, calculated by the following formula: $v = \sqrt{h} \times 66{\cdot}1$, is 136·7 feet per second or 8,202 feet per minute, while the periphery speed of the fan is only 6,573 feet per minute. It is extremely probable, if a lower velocity of blade-tip speed can raise a depression producing a higher speed of air, that there are some deeper problems underlying the action of centrifugal fans than have been thought of, and the whole theory of fan action may have to be revised.

The object of the writer of this paper is to show that high manometric effect does not necessarily imply high useful effect, and also to point out the need of some accepted rules for testing fans, to avoid unsafe results in manometric effect and useful effect in comparing fans, where the water-gauge has been placed under entirely different conditions.

Further trials were made on May 23rd last, of the Guibal and Capell fans at the Marles collieries, in the north of France. The results were :—

	Guibal Fan.	Capell Fan.
Fan, diameter (feet)	23 ...	12½
Revolutions per minute, engine ...	75 ...	63
Do. do., fan	75 ...	306
Water-gauge, near inlet (inches) ...	1·40 ...	9·10
Do. in main drift in a recess, and common to both fans (inches) ...	0·95 ...	8·35
Volume of air per minute (cubic feet)	72,559 ...	203,328

The difference of 0·45 inch between the inlet and static water-gauge in the case of the Guibal fan is very large, while the difference between the inlet and static water-gauge in the case of the Capell fan is only 0·75 inch. The efficiency of the Guibal fan is very low, being only 39 per cent. of the inlet water-gauge.

The PRESIDENT said he thought it would be desirable in discussing this paper, to take at the same time the discussion on the following papers :—"Observations on Fans of Different Types Working on the same Upcast Shaft," by the Rev. G. M. Capell;* and "Experiments upon a Waddle Fan and a Capell Fan Working on the same Mine at Equal Periphery Speeds at Teversal Colliery," by Mr. J. C. B. Hendy.†

Mr. W. COCHRANE said he would like Mr. Capell to make his meaning perfectly clear as regards the manometric efficiency. In the paper it was said to refer "simply to the relation of the water-gauge to the speed of the blade-tips of the fan, and had no necessary relation to the useful effect of the fan." Did this mean that the manometric efficiency was the water-gauge (irrespective of where it was taken) compared with the theoretical water-gauge? If so, it was simply the relation of the indicated water-gauge (the water-gauge itself, measured and seen) to the theoretical water-gauge; and he wanted to know whether the latter was based upon $h = u^2 \div g$ or $h = u^2 \div 2g$?

Mr. A. L. STEAVENSON referred to the same paragraph as to the manometric efficiency in which Mr. Capell said "it refers simply to the

* *Trans. Fed. Inst.*, vol. iv., page 203. † *Ibid.*, page 474.

relation of the water-gauge to the speed of the blade-tips of a fan, and has no necessary relation to the useful effect of a fan." If they took any ventilator and found that some part of its work was performed more efficiently than in any other fan, it followed that the useful effect, all things being equal, would also be higher. Mr. Thomas Bell (H.M. Inspector of Mines) and Mr. Cooke were the first to make observations upon the pseudo water-gauge, and they noted that the water-gauge in the case of the Guibal fan had in all cases exceeded $u^2 \div 2g$, which, he thought, was accounted for by the action of the évasée chimney. Undoubtedly the évasée chimney entirely altered the conditions under which the speed of the blade-tips governed the theoretical water-gauge. The general law of the manometric efficiency of fans was now so well established that it could not be resisted; in fact, Mr. Capell had recognized the advantage of the évasée chimney, and now used it in the construction of his fan. The only question now in dispute was the position of the water-gauge. He thought the proper position was at such a distance from the fan that it was not affected by the speed of the fan, say, for instance, at the bottom of the pit.

Prof. J. H. Merivale said he understood Mr. Capell to say that it should be taken as near to the fan as possible, and where the air was practically at rest. The best position would be in the fan-chamber.

Mr. Capell said with regard to the question of the higher manometric efficiency necessarily implying a higher useful effect, his own experience was diametrically opposed to such a conclusion, and in the recent experiments at Marles colliery, carefully conducted by capable engineers, they arrived at a result which was a startling surprise.

Mr. W. Cochrane asked the meaning of "double centrifugal type of mine-ventilator," and whether it applied to others than the Capell fan? Would he call the Guibal or Waddle fans "double centrifugal"?

Mr. Capell said they were single-centrifugal fans, but the Pelzer and Rateau fans were of the double centrifugal type.

Mr. W. Cochrane referred to Mr. Capell's statement that "some fixed distance from the inlet should be settled for the water-gauge tube-end, so as to avoid conclusions which are misleading and inaccurate." If he (Mr. Cochrane) remembered rightly, Mr. Capell argued in a previous discussion* that the principle of the Capell and similar fans was such that they carried back (for the same velocity of blade-tip) a water-gauge very much farther into the mine than any other kind of fan; therefore, if at the surface, a distance of 20 feet or so away, were settled as a

* *Trans. Fed. Inst.*, vol. iv., page 204.

suitable position for the water-gauge, an advantage would be claimed in all cases for Capell fans and those of similar type; that is, that they would be able to indicate a higher water-gauge at a distance of 20 feet from the fan than any other fan. The Guibal fan at Marles, which produced 72,559 cubic feet at 0·95 inch water-guage, in the fan drift would produce 203,000 cubic feet in the same mine at 7·436 inches of water-guage. It seemed to him that the Capell fan indicated a higher water-guage to circulate this volume than the Guibal fan, but such higher water-guage was not required, and ought to yield, if utilized, a much larger volume of air than 203,000 cubic feet per minute.

Mr. Capell asked what would have been the water-gauge of the Guibal fan when circulating 203,000 cubic feet per minute?

Mr. Cochrane—In the proportion of $72^2 : 203^2 = 0{\cdot}95 : 7{\cdot}436$ inches.

Mr. A. L. Steavenson said that if they took the experiment on the Guibal fan and used the formula $u^2 \div 2g$, they got an efficiency exceeding 100 per cent., which was impossible. It was necessary to recognize the effect of the évasée chimney, and the square of the speed of the periphery should be divided by g and not $2g$.

Mr. T. H. M. Stratton said he would leave the scientific part of the discussion to Mr. Cochrane, but he thought Mr. Capell was under a mistake if he assumed that the manometric efficiency had anything to do with the useful effect. If they had a fan perfectly suitable to the size of the mine, running at a certain speed, and with its outlets and intakes adapted to that mine, they obtained a high useful effect, but they could not predict the proper size of fan to put up unless they knew the area of the orifice of the mine.

Mr. W. C. Blackett said he was in considerable doubt as to what Mr. Capell meant to convey by his paper, more especially as to water-gauges. From his use of the term "static" as applied to a water-gauge in one part of his remarks, and in another to some mysterious influence "thrown back from the fan," he (Mr. Blackett) was not able to say whether the difference was due to this mysterious influence "thrown back," or to the dynamical effects of the air-currents. As Mr. Capell used the word "static" he thought he had in mind the dynamic effects of air-currents passing the open mouth of a tube connected with the water-gauge; this was one thing, but this influence "thrown back" from the fan was another, and he (Mr. Blackett) could not quite understand whether in speaking of having to be careful against the static gauge they were to be careful to get out of this mysterious influence or to get

out of the influence of the dynamic effects of air-currents. In the latter case, he did not see how putting the water-gauge on the second of the separation-doors got over the difficulty, as in that case they simply altered the shape and length of what might be called equivalent to the tube of the water-gauge. He understood Prof. Merivale to say that it was not necessary for the air in connexion with the water-gauge to be perfectly at rest, but Mr. Capell considered that it should be at rest. But he (Mr. Blackett) took it that in all water-gauges connected by the fan-drift with the tube, the air in the tube was at rest, and merely altering the shape of the end of the tube and expanding it into a passage at right angles—or any other angle—to the fan-drift did not provide them with air not in motion, and he thought in fact the widening out of the tube and placing the widened mouth against the air-current might prevent them getting much water-gauge at all. About fifteen years ago he made some experiments upon a Schiele fan with the special object of getting to know the influence of the position and direction of the pipe upon the readings of the water-gauge, in relation to the air-current and the orifice of the fan, and he was able to get from 0·30 inch with the pipe-mouth facing the current to over 2 inches where the pipe was reversed, and there was at the same time about 0·50 inch registered on the drift doors. Although Mr. Cochrane might be in doubt as to which theoretical water-gauge ought to be accepted, there was considerably more doubt as to how they were to measure the practical water-gauge, and he had not heard anyone yet give any rule as to taking a water-gauge that would be equally applicable to any fan-drift. He thought they would always have differences in their observations—one gentleman on one side doubting the veracity of another gentleman on the other side as to the results produced from their respective fans—unless they could hit upon some contrivance, be it simple or otherwise, for getting a water-gauge free from all influence of air in motion, and especially free from that mysterious influence which Mr. Capell said was capable of being "thrown back" from the fan.

Mr. A. L. Steavenson said he thought he could elucidate the matter. The water-gauge decreased as they got farther away from the fan, until they reached a point at which it became practically constant. If a fan created a vacuum giving more air than the fan could contain, they would get a pseudo water-gauge. A large fan of 40 or 50 feet in diameter produced no pseudo water-gauge; but a small fan running on a large mine produced great pseudo water-gauge, because the vacuum produced more air out of the mine than the fan could swallow.

Mr. W. C. Blackett rejoined that Mr. Steavenson's statement proved that he was more influenced by the mysterious influence of the fan "throwing back" than by the dynamical effects of the air-current, and he (Mr. Blackett) was still in as much doubt as ever as to which should have the more importance attached to it.

Mr. Capell said that in order to avoid all dynamic influence of the current on the water-gauge, he had tried the experiment of using a sharp pointed tube, which he drove into a ball of worsted where nothing could get at it. For practical use a tube of perforated zinc placed on the end of the water-gauge tube, and covered with cotton-wool and then wrapped over with several layers of flannel was equally efficient.

Mr. T. H. M. Stratton said that roughly speaking, they might put the manometric efficiency of the Guibal fan at 0·64 and the Waddle, Schiele, and Capell at 0·46.

Mr. Ferrier said he did not think the manometric efficiency affected the mechanical efficiency, for the highest manometric efficiency was obtained when the mine was shut off, when the mechanical efficiency was *nil*. He did not see how the dynamic effects could very readily be produced in a substance of such small density as air; in hydraulic experiments they had a very great effect on the pressure at a particular point along the pipe through which the water was passing, but he thought this was very different to what the effect would be on a substance of such small density as air. The energy must be very small in consequence of the small weight of air, and it was difficult to see how the dynamic effect could produce an appreciable effect on the water-gauge. They might suppose that the pressure would form rapidly as it went along the drift to the end of the fan, but it would be difficult to find at what part it would be best to place the water-gauge; it would depend largely on the passage through which the air was moving; the larger the area through which the air was passing the less the pressure required to drive it through.

Mr. Capell said if they took a glass tube 3 feet long and blew across its end they would get a water-gauge of 14 or 15 inches, and this was exactly what happened in a tube placed in a very high velocity of air-current; the air whistling past its end raised the observed water-gauge to a higher point than the static water-gauge would be.

Mr. J. C. B. Hendy said that he agreed with Mr. Capell's remarks as to the dynamic results observed in the drift. The dynamic water-gauge would be due to the velocity or impulse of the air passing the orifice of the tube; that is to say, if the orifice of the water-gauge tube was uncovered in one instance and covered in another, there would probably be a

difference in the water-gauge, and that difference would be due to the dynamic effect. He understood Mr. Capell to say, in the paper he had just read, that a fan might have a very high manometric efficiency and at the same time a low mechanical efficiency, and that one machine might be a better ventilator with a high manometric efficiency and a low mechanical efficiency, than another machine which gave a comparatively low manometric efficiency and high mechanical efficiency. This raised the question as to whether they were to judge the fan and engine as a whole, or one machine, or to separate the two and simply judge the fan from its manometric efficiency and the volume of air it produced. No doubt the fan might suffer in the comparison if driven by a badly constructed or inefficient engine, but on the other hand they ought to know which fan required the most engine power, and he thought that experiments ought to be made so as to obtain such data as would enable them to make a fair all-round comparison. In most of the cases quoted by Mr. Capell where a Guibal or other large fan had been replaced by a Capell fan, it should not be forgotten that the former had been replaced by a Capell fan designed to give a much larger volume of air than the old fan was capable of exhausting, and (in the comparisons he makes) he gives the calculated water-gauge required for the old fan against the actual water-gauge obtained with the new fan. He did not think it was right to compare the calculated water-gauge from a fan designed to produce say 50,000 cubic feet per minute with the actual water-gauge given by a new fan designed to produce 120,000 or 150,000 cubic feet per minute. What they wanted was a comparison between fans each designed for the same duty and working under the same conditions, and so far as he could remember Mr. Capell had not quoted any such instance.

Mr. M. Walton Brown said that Mr. Guibal (many years ago) showed that in either a cased or open-running fan a force equal to that induced by the velocity of the circumference was wasted, and he further proved that the ideal centrifugal fan should be provided with an expanding-chimney or similar apparatus which would utilize the work remaining in the air on its issue from the fan. If that adjunct was of infinite dimensions, Mr. Guibal showed that the water-gauge would be doubled, and it had not been found difficult in practice to make a centrifugal fan, provided with an expanding-chimney, produce 80 to 83 per cent. of the theoretical yield calculated by the formula $u^2 \div g$. In other words, Mr. Guibal accepted $u^2 \div 2g$ as the value of the theoretical water-gauge produced by either cased or open-running centrifugal fans; and in the case of the ideal centrifugal fan provided with sliding-shutter and expanding-chimney, he proved that this value of the theoretical water-gauge was

doubled, or 2 $(u^2 \div 2g) = u^2 \div g$, which agrees with Mr. Murgue's formula. He (Mr. Brown) could not accept Mr. Capell's view that the manometric efficiency of any type of fan could be considered as having a constant fixed value, irrespective of the conditions of the mine. The following table records the work of the old single open-running inlet fan at Birch Coppice colliery,* together with certain calculated results deduced

No. of Experiment.	Observations.			Calculated Results at 106·1 Revolutions per Minute.			Squares of Volume. (v^2.)	Water-gauge.		Ratio of Observed (h) to Theoretical Water-gauge ($h' = 1{\cdot}968$ in.).
	Revolutions per Minute.	Volume of Air per Minute.	Water-gauge.	Equivalent Orifice.	Volume of Air per Minute. (v.)	Water-gauge. (h.)		Calculated. (h'.)	Difference. ($h - h'$.)	
		Cubic Ft.	Ins.	Sq. Ft.	Cubic Ft.	Inches.		Ins.	Ins.	
1	152	70,280	1·10	24·75	49,057	0·536	2,406,599,249	0·577	−0·041	0·269
2	120	60,000	0·70	26·53	53,050	0·547	2,814,302,500	0·544	0·003	0·275
3	137	73,000	0·90	28·47	56,535	0·539	3,198,206,225	0·513	0·026	0·271
4	137	73,000	0·85	29·30	56,535	0·509	3,198,206,225	0·513	−0·004	0·256
5	152	118,498	0·45	65·35	82,714	0·219	6,841,605,796	0·216	0·003	0·110

therefrom. If a diagram be made with the squares of the volumes (v^2) as abscissæ, and the observed water-gauges (h) as ordinates, a straight line could be drawn approximately through the five points, having for its equation $h = 0{\cdot}773$ inch $- 0{\cdot}000{,}000{,}000{,}081{,}5\, v^2$. The calculated water-gauges (h') are obtained from this formula, and the difference ($h - h'$) shows the apparent errors. These differences are slight, except in the case of No. 1 experiment, and may be set down to errors of observation. Taking the initial water-gauge (0·773 inch) and the observed water-gauge (h), it will be seen that the manometric efficiency is not constant for the Birch Coppice fan, and that it varies from 0·462 at the initial water-gauge, to 0·110 in No. 5 experiment. Mr. Capell, in a previous paper,† alleges that there is "a different water-gauge in the Guibal fan when passing the same volume of air as the small Capell fan," and "the lower gauge for a given work in air shows itself wherever the Capell fan has replaced a Guibal fan." The statement is not borne out by the experiments recorded in Mr. Capell's paper, as the comparison is made by calculation of the results of the Guibal fan at a low peripheral speed with those of the Capell fan at a much higher peripheral speed, and not by direct comparisons of the water-gauges of both fans when actually passing the same large volume of air. Thus at Prosper I collieries, the Guibal fan producing a volume of 59,340 cubic feet is compared by calculation with

* *Trans. South Staffs. Inst. Min. Eng.*, vol. xi., page 74.

† *Trans. Fed. Inst.*, vol. iv., page 203.

the Capell fan volume of 127,574 cubic feet per minute; and at Marles, the Guibal fan exhausting 72,559 cubic feet is compared with a Capell fan producing 203,328 cubic feet per minute. The manifest unfairness of such comparison of the results obtained by one fan with those produced by another fan running at a much higher peripheral speed is clearly evidenced by a comparison of Nos. 1 and 5 experiments upon the Capell ($12\frac{1}{2}$ feet double-inlet) fan at Silverhill colliery,* as follows:—

	No. 1 Experiment.		No. 5 Experiment.
Revolutions of fan per minute	123	...	215
Volume of air per minute (cubic feet) ...	110,920	...	199,550
Water-gauge in fan-drift (inches) ...	1·40	...	4·10

Taking No. 1 experiment of 110,920 cubic feet under 1·40 inches of water-gauge, the Capell fan would require not less than 4·50 inches of water-gauge instead of 4·10 inches, actually observed in No. 5 experiment. These experiments show that a fan (in certain cases) yields better results than a fan running at a lower speed, and that such an unfavourable comparison at a low speed occurs equally in the case of the Capell fan as in any other fan. He (Mr. Brown) had considered the experiments made upon the Capell and Waddle fans at Teversal collieries, and he thought that the following experiments were worthy of comparison:—

Description of Fan.	No. of Experiment.	Diameter.	No. of Revolutions per Minute.	Volume of Air per Minute.	Water-gauge.	Reference.
		Feet.		Cubic Ft.	Inches.	
Capell ...	A_1	16	113	88,045	1·40	*Trans. Fed. Inst.*— vol. iv., page 476.
Waddle ...	A_2	30	52	87,729	1·00	vol. iv., „ 477.
Capell ...	B_1	16	131	101,747	1·90	vol. iv., „ 476.
Waddle ...	B_2	30	65	103,381	1·95	vol. ii., „ 543.
Capell ...	C_1	16	135	116,494	2·20	vol. iv., „ 477.
Waddle ...	C_2	30	70	114,309	2·15	vol. ii., „ 543.

The volumes in the above table, when reduced to a peripheral speed of fan of 6,000 cubic feet per minute, show the following comparative results:—

No. of Experiment.	Capell Fan.	Waddle Fan.
	Cubic Feet.	Cubic Feet.
A	93,000	107,400
B	92,700	101,200
C	103,000	103,900
Mean ...	96,200	104,200

* *Trans. Fed. Inst.*, vol. ii., page 549.

These experiments also prove that the Waddle fan produced practically the same volume of air at a much less water-gauge than the Capell fan at Teversal collieries.

Mr. CAPELL said that in considering the output of fans, the cubic contents of the fan and the output per revolution should not be forgotten. The Capell fan at Marles collieries had more than double the output per revolution of the Guibal fan. Working under the very high water-gauge required at Marles collieries, he calculated that 208,000 cubic feet would be exhausted at 310 revolutions per minute under 9 inches of water-gauge, and, as a matter of fact, at 306 revolutions the quantity of air was 203,328 cubic feet per minute under 9·10 inches of water-gauge, which was very near to the difficult guarantee which he had given. The rule for calculating the theoretical water-gauge referred to in his paper was Mr. Murgue's rule, $h = u^2 \div g$. On the Continent, this rule is now universally acknowledged, and the manometric efficiency is measured by its results. He (Mr. Capell) did not advocate taking the water-gauge close to the fan, as it was then influenced by local effects. Where practicable, it should be taken at a distance of 20 feet, and in a recess. He was questioned with regard to higher manometric efficiency implying higher useful effect; in the December (1892) tests of the Marles fan,* the manometric effect with a less volume of air than was obtained in May last was 0·547, with a mechanical efficiency of 0·578. A new form of shutter had been placed in the fan, and the chimney water-gauge had been decreased by its use. The result showed a corresponding decrease in the manometric effect, as in the last trials it was only 0·5109, but the useful effect rose to 0·591. The rise being about the same as the decrease in the manometric efficiency. The 12 feet single inlet fan, to which Mr. Brown refers, was the second Capell fan used on a colliery, and was an open fan of a type now discarded, and its results were not constant. The fan was an experimental one, from which useful deductions for future work were obtained. He (Mr. Capell) strongly protested against there being any unfairness towards a fan of 40 feet diameter and 10 feet wide at Prosper colliery being compared with a fan 12 feet 6 inches in diameter and 6 feet wide, or a fan 23 feet in diameter at Marles colliery being compared with another 12 feet 6 inches in diameter. The comparison is fair, as the large fans working on low water-gauges get the advantage of natural ventilation, which passes away when producing water-gauges of 6 inches to 11 inches. The unfairness, if any, seemed to be all the other way. Two independent observers have made tests of the Teversal fans, Mr.

* *Trans. Fed. Inst.*, vol. iv., page 215.

W. Piggford and Mr. Maurice Deacon. They have both proved the abnormal resistance required to get the air to the Capell fan inlet. Both of them have found that at the same water-gauge taken on the same water-gauge pipe for both fans on the drift, and in spite of the resistance between drift and inlet, the Capell fan has the advantage both in volume and useful effect. The water-gauge to get the air to the inlet from the drift is work done needlessly, and with a better arrangement the Capell fan would use that water-gauge in getting large volumes of air. As so much discussion had been raised over the trials of the fans at Teversal collieries, he hoped it might be possible to test the fans under nearly the same conditions by placing a temporary arrangement in the drift (if permission could kindly be given); but it would be quite impossible to get exactly the same conditions owing to the peculiar position of the Capell fan which is laid across the drift. Mr. Cochrane in his calculation had taken the static water-gauge as the water-gauge of the Guibal fan, instead of the inlet water-gauge, and had left out the effects of an important factor, the natural ventilation of the mine, which would come into action very strongly at the low water-gauge of 1·40 inches, in reducing the proportion between the inlet water-gauge and static water-gauge, but would not influence the high water-gauges. As a matter of fact there was a natural ventilation of about 16,000 cubic feet per minute at Marles colliery, and those who have studied the recent paper of Mr. T. A. Southern* will at once recognize how unsafe it is merely to rely on the squaring of the air-volume to get the water-gauge required for a large increase of the ventilation, unless natural ventilation is allowed for and a corresponding proportion of water-gauge added to the direct calculation from the squares of the volumes. In the Marles experiments, the 23 feet Guibal fan gave a useful effect of 39 per cent. on the water-gauge measured in the inlet of the fan, and 27·9 per cent. on the static water-gauge taken in a deep recess in the wall of the main-drift, which was 140 square feet in area. Hence, naturally, the static water-gauge useful mechanical effect of the fan had not been noticed in previous trials of this Guibal fan. The Guibal fan produced in round figures 72,000 cubic feet per minute at a water-gauge of 1·40 inches in the inlet to the fan; the static water-gauge for 203,000 cubic feet by calculation is obtained thus: As $72{,}000^2 : 203{,}000^2 = 1{\cdot}40 : 11{\cdot}109$ inches. The horse-power in the air is found to be: $203{,}000 \times 1{\cdot}40 \times 5{\cdot}20 \div 33{,}000 = 355{\cdot}5$ horse-power; and at 39 per cent of useful effect, the Guibal fan would have required $355{\cdot}5 \times 100 \div 39 = 911$ horse-power to have produced 203,000

* *Trans. Fed. Inst.*, vol. iv., page 460.

cubic feet of air per minute. The Capell fan results previously recorded show 291·5 horse-power in the air by water-gauge near inlet, and of 267·5 in the air by the static water-gauge; the engine being indicated at 495·0 horse-power; the useful effects being 59·1 per cent. by the water-gauge taken near inlet, and 54 per cent. by the static water-gauge. The contrast thus worked out is very clear, and the efficiency of the Guibal fan, worked out both on the inlet water-gauge and the static water-gauge, is also evident. The manometric efficiency of the Guibal fan at 75 revolutions, taking the inlet water-gauge, is 38·5 per cent., and taking the static water-gauge it is 26·12 per cent. The Capell fan, at 306 revolutions, gives on the water-gauge near the inlet 51·09 per cent. of manometric efficiency, and on the static water-gauge gives 46·8 per cent. of manometric efficiency.

The PRESIDENT said the discussion would now be adjourned, but in the meantime he thought their hearty thanks were due to Mr. Capell for endeavouring to enlighten them on these difficult problems.

The proposal was cordially adopted.

Mr. CAPELL responded to the vote of thanks.

DISCUSSION UPON PROF. F. CLOWES' PAPER ON "A PORTABLE SAFETY-LAMP, WITH ORDINARY OIL ILLUMINATING FLAME, AND STANDARD HYDROGEN-FLAME, FOR ACCURATE AND DELICATE GAS-TESTING."*

In the absence of Prof. Clowes, Mr. PALMER exhibited and explained the use of the hydrogen-oil safety-lamp.

The PRESIDENT asked if the lamp was in actual use?

Mr. PALMER replied that it had been in actual use for some months at the West Riding collieries.

Mr. A. L. STEAVENSON suggested that if the appliance would give equal results in ordinary coal-gas they might see some experiments with it now.

Mr. PALMER said this was not practicable, owing to the absence of a test-chamber.

Mr. M. WALTON BROWN emphasized the remarks he made when the lamp was shown at Derby,† and said that he would like information on

* *Trans. Fed. Inst.*, vol. iv., page 441. † *Ibid.*, page 456.

the following points :—(*a*) Was there any danger, when filling the hand cylinder with hydrogen from the large cylinder, of an explosive mixture being formed ? If the valve of the hand cylinder was left open it might be filled with air, which would become explosive if mixed with hydrogen from the large cylinder, after the pressure had been reduced by repeated chargings of the hand cylinder. (*b*) The risk of the hydrogen being turned into the lamp at full pressure, and burning a hole in the gauze or cracking the glass. (*c*) The risk of the passage of flame through the pipe used for the introduction of the hydrogen gas from the hand cylinder. Prof. Clowes stated that* "It should be noted that the hydrogen-flame can be set to its standard height in the presence of the gas in measuring the flame-caps, and that the test can therefore be at once made in the air to be examined without previously placing the lamp in gas-free air." He thought that if, say, 1 per cent. of gas was mixed with the air, the bulk of the hydrogen-flame would be enlarged accordingly, and it seemed to him impossible to adjust the hydrogen-flame to its standard height in such a mixture of gas and air. He thought the use of the lamp might be greatly simplified, if the gas could be produced chemically by an attachment to the lamp. He had used the hydrogen produced in a Döbereiner lamp as a test-flame for fire-damp, and suggested that Prof. Clowes might with advantage adopt a similar system.

Mr. Palmer, referring to the question of air being left in the cylinder, said that no harm would result, and two or three chargings would entirely remove it. He thought the lamp would be safe if the hydrogen were suddenly turned on at full pressure; the flame would simply shoot to the top of the lamp, but he did not think it would burn a hole through the gauze before being reduced.

Prof. Merivale observed that this point could be easily settled by experiments.

Mr. Palmer, continuing, said the hydrogen-flame was always of the same height, and there was a clear line of demarcation between the hydrogen-flame and the gas-cap, which were perfectly distinct.

Correspondence.

Dr. P. P. Bedson wrote regretting his absence during the discussion on the hydrogen-flame lamp, for he would have wished to express his admiration of the ingenuity displayed in the construction of the lamp. At the same time, he (Dr. Bedson) must confess to a feeling of disappoint-

* *Trans. Fed. Inst.*, vol. iv., page 451.

ment at the indications yielded in low percentage mixtures of fire-damp and air; in fact, in those ranges where an indication is most needed. The difference in the length of flame in $\frac{1}{4}$ per cent. mixture and $\frac{1}{2}$ per cent. mixture is practically useless as a means of gauging the amount of gas present, and does not compare at all favourably with the results of experiments with the Pieler lamp, made by the Austrian Fire-damp Commission, or of those conducted by Dr. Brookmann for the Prussian Fire-damp Commission. Dr. Brookmann states the difference in height of flame to be 2 centimetres (0·79 inch), which is about twenty times the difference in height of the hydrogen-flame. The Austrian Commission found a similar difference in the height of the flame, when experimenting with mixtures of Rossitzer gas, Segen Gottis mine, corresponding to $\frac{1}{4}$ and $\frac{1}{2}$ per cent. of marsh gas; whilst with other natural gases—the same difference in amount of marsh gas—the flames differed in height by 1 centimetre (0·39 inch), which is still ten times greater than the difference observed with the hydrogen-flame. These facts show the hydrogen-lamp cannot for such low percentages compare for delicacy with the Pieler lamp as an indicator of fire-damp. He (Dr. Bedson) would like to have asked Prof. Clowes how, when the lamp has once given an indication of gas, it is possible to set it without again bringing the lamp into an atmosphere free from gas. He (Dr. Bedson) understood that one of the disadvantages under which the Pieler lamp labours arises from the volatility of the alcohol; but the construction of the hydrogen-lamp requires the presence of a volatile oil, which is a factor of danger, just as alcohol is with the Pieler lamp. It would appear desirable that experiments with all such lamps should be made not only with marsh gas, prepared in the laboratory, but with gas collected in the mine, for, as shown in the experiments of the Austrian Commission, considerable differences may be observed, and it would be useful to know how these fire-damp indicators behave in different samples of fire-damp.

Dr. Clowes wrote that he fully endorsed the replies given by Mr. Palmer. In practice the small cylinder was found to be freed sufficiently from air by charging it once with hydrogen, and allowing this to blow off. Those who had used the lamp for some time underground, made no complaint of any difficulty in regulating the supply of hydrogen so as to avoid the production of an unduly large flame. The passage of flame outwards from the interior of the lamp through the hydrogen tube had been proved by many experiments to be impossible. Great importance is attached to the fact that in measuring the cap-heights, the hydrogen flame was always set to its standard height in air containing the known

percentage of gas. It is therefore unnecessary to remove the lamp to air free from gas in order to adjust the standard hydrogen-flame. Unfortunately the chemical generation of hydrogen gas from materials within the lamp had been long ago proved to be an impossibility. The flame of the gas chemically produced could not be regulated, and the metallic parts of the lamp were constantly corroded by the chemicals used. An experimental lamp, with which the attempt was made, never left the chemical laboratory. On the other hand, the present lamp, in which the hydrogen was prepared, and then applied in a compressed state, had, after lengthy trial worked very satisfactorily. He (Dr. Clowes) regretted that Dr. Bedson had not had an opportunity of seeing the hydrogen-oil lamp exposed to low percentages of gas in the test-chamber and in the pit. If this opportunity had been afforded (and Dr. Clowes was now prepared to offer it in any convenient way or time), no doubt would have been expressed as to the indications furnished by the lamp in $\frac{1}{2}$ and even in $\frac{1}{4}$ per cent. of gas. It would be seen that the cap-heights for these low percentages of gas ($\frac{1}{2}$ and $\frac{1}{4}$) were practically equal, but that the caps were distinguishable even by a novice without the slightest hesitation. The $\frac{1}{4}$ per cent. cap is hazy, pale, and indefinite in outline, but perfectly visible, and the $\frac{1}{2}$ per cent. cap is much less pale, and is comparatively well-defined. The caps are not distinguishable by difference in height, but a glance serves to distinguish them by their appearance. As regards the distinction of the different percentages, from $\frac{1}{4}$ to 6, no difficulty has presented itself either in the laboratory tests in the test-chamber or underground, in the hands of officials. It is true that the caps are all smaller than those shown by the Pieler lamp, but they are quite large enough to give easily appreciable differences to the unaided eye, and when viewed against the special gauge in this lamp they can be noted with certainty by any one. The Pieler caps are unnecessarily large, and are very seriously hindred from being properly seen by the fact that they are viewed through metallic gauze, and with a most unsuitable gauze background ; whereas in the hydrogen-oil lamp the caps are viewed through transparent glass against a dead-black and most suitable background. The standard hydrogen lamp could be set to its standard height with absolute certainty in any part of the pit, and did not require to be taken into fresh air for adjustment. This had been secured by always setting the flame to its standard height in the presence of the gas in the test-chambers, and then reading the cap-height which was registered on the gauge and the scale-card for use in the pit. For low percentages of gas this was, however, unimportant, since the hydrogen-flame, when set

in air free from gas, did not appreciably increase in size until more than 3 per cent. of gas was present in the air. For detecting percentages of gas above 3, the use of the reduced oil-flame (also set in the presence of the gas) was recommended. Dr. Bedson was in error in stating that a volatile oil was used in the hydrogen-oil lamp. The oil burnt was a mixture of colza with water-white paraffin ; the latter was not volatile, and was considered to be a perfectly safe oil in any form of lamp. The paraffin was introduced to prevent charring and crusting of the wick, and thus to improve the burning and the ease of reduction of the flame by drawing down the wick, it also improved the light of the flame. As regards the testing of the lamp in mine-gas as well as in artificially prepared marsh-gas, it should be remembered that in the preparation of a scale for the lamp, known percentages of marsh-gas must be employed, and this is most readily and certainly secured by mixing methane with air in known proportions. Carbonic acid is the only gas likely to affect the cap, and special experiments in the test-chamber have shown that even 5 per cent. of carbonic acid does not affect the cap. The precaution, however, has been taken of making tests side by side in the main return airway with the hydrogen-oil lamp, the Pieler lamp, and the Liveing indicator, with the result that, in percentages of gas ranging from $\frac{1}{4}$ to 2, the indications of the three instruments were absolutely identical. Dr. Clowes wished further to state that in comparative tests in low percentages of gas made with the hydrogen lamp and with the Pieler lamp, the caps were easily read in the hydrogen lamp, but they were not readable in the Pieler lamp furnished by an English firm. This defect was partly due to the luminosity of the large alcohol flame, but largely to the interference caused by the metallic gauze. The brass gauze partly acted by impeding the passage of the light of the faint caps, but mainly by its reflecting power. Some improvement was noticed when the gauze was blackened. But only on the removal of the bonnet and gauze were the tall caps satisfactorily seen in their full dimensions.

The President proposed a vote of thanks to Mr. Palmer for attending to explain the use of the testing-lamp. He hoped something

practicable would before long result from the various instruments which were brought forward from time to time, and this lamp certainly seemed to tend towards the object they wished.

The vote of thanks was unanimously adopted.

The following paper was taken as read :—"The Choice of Coarse and Fine-crushing Machinery and Processes of Ore Treatment," by Mr. A. G. Charleton.

THE CHOICE OF COARSE AND FINE-CRUSHING MACHINERY AND PROCESSES OF ORE TREATMENT.*

By A. G. CHARLETON.

Part III.—Silver.

Wet and Dry Pan-amalgamation and Lixiviation.

Pan-amalgamation, as applied to silver ores, is always preceded by stamping the ore in a battery. If conducted wet, the pulp is collected in tanks, from which it is shovelled into the pans (an arrangement which might, the writer thinks, be improved upon), or else, if arranged on the Boss system (much in fashion lately owing to the saving in labour and other advantages claimed for it) the pulp is run straight through a series of pans without any intermediate settling. If conducted dry, the ore is taken direct from the cooling-floor of the roasting furnace (which forms an essential part of the plant), and is charged into the pans afterwards.

Pan-amalgamation is most extensively used in the Western States of America for the extraction of silver from its ores, and with such exceptions as have been or will be alluded to, is, the writer thinks, likely to retain its position for some time to come.

It broadly divides itself, as will be presently seen, into:—

1. The Washoe process, in which the ores go direct from the tanks to the pans wet, the amalgamation being generally assisted by the use of chemicals, chiefly salt and bluestone.
3. The Reese River process, in which the ore must be first dried: (*a*) on a drying-floor heated by waste-steam or furnace gas; or (*b*) in revolving drying-cylinders or shelf-kilns, which are steadily coming more into fashion. Roasting (usually with salt, which is mixed in the battery, or between it and the furnace) follows drying, and the ore is ground in pans in the same way as in the wet process.
3. The continuous Boss process.

* *Trans. Fed. Inst.*, vol. iv., pages 233 and 351.

The Washoe Process.

In an ordinary wet-crushing silver mill, the ore is brought in cars to the top of the mill-building, where it is dumped over the top of the inclined grizzley or screen on to the crusher-floor. All the small pieces pass through the grizzley into the ore-bins below. The coarse rock is shovelled into the jaws of the rock-breaker, which are on a level with the crusher-floor. The ore crushed to walnut size in passing through the rock-breakers falls into the ore-bins, and thence goes to the automatic feeders (behind the stamps), passing through inclined shoots controlled by gates. The automatic feeders being kept full, ensure a uniform supply of stone being fed to the stamps as fast as needed.

The finely stamped ore, known technically as pulp, suspended in water, flows into large settling-tanks, where the excess of water is drawn off, while the thick pulp remaining is shovelled in regular charges into a row of amalgamating-pans, in which it is ground for several hours, first with salt, bluestone, and other chemicals, and afterwards amalgamated with mercury, with the mullers raised. The contents of the pans are run into large settlers (when the previous operation is finished) placed below, and in front of the pans, in which the pulp is thinned by additions of water and gentle agitation, and all the quicksilver with the precious metals in the form of amalgam, settles to the bottom. The pulp is gradually drawn off from the settlers (through holes fitted with plugs at different levels in the side) and flows to waste. The amalgam is strained from the excess of quicksilver, retorted to drive off what remains, and the resulting gold and silver cake is melted into bars. The gold and silver contained in the sulphides, which will not yield to the above treatment, is sometimes caught by concentrators (Frue or Embrey) which receive the waste pulp-tailings from the settlers. A clean-up pan generally forms part of the plant.

The old method usually employed in silver-milling was to crush coarse in the battery and grind fine in the pan, but as this involves greater power, greater wear of castings, greater loss of mercury, and not always better results, the system of crushing fine in the battery (keeping the shoes barely off the dies) has lately come into practice. By this means more gold and silver will sometimes be extracted, since ore will naturally break where there is most mineral, and the fine comminution in the battery will generally disengage most of it, the consequence being, that as quicksilver has a preference for gold and silver, it will amalgamate with them, rather than take up base metals (which render it inactive), which it is forced to do by excessive grinding.

The modern amalgamating-pan is a growth from the old arrastra, and though its construction is quite simple it presents a variety of forms. The pan holds from 1 and $1\frac{1}{2}$ to 2 tons of pulp, and generally revolves at about 60 revolutions per minute; the gearing underneath is open and plain, the muller is raised by a left-hand screw on top, the hand-wheels of which should be large (the jam-wheel being no smaller than the screw-wheel), as it frequently requires a greater application of muscle than the latter, and when the machinery is in fast motion it is inconvenient to adjust a small wheel under a large one. The most important feature of the pan is the pulp-current, which often receives but little attention, and though simple in principle is not always understood, and its neglect may cause serious loss.

These currents must be uniform and regular to ensure uniform work, and strong enough at the bottom of the pan to carry the quicksilver. The motion of the muller makes a current by throwing the pulp to the outside as it advances, which then rolls up at the side and falls over towards the centre, and down through the central opening, in and under the muller, to be thrown outwards again from the bottom. This so far cannot be improved upon, but the wings are needed, to give the pan capacity (by preventing the pulp from running too high up at the side), to accomplish which, they should have the shape of an inverted ploughshare.

Having naturally a good current above the muller, we have only to work in unison with that underneath it, which will depend on the design of the muller and setting of the dies.

The pans vary in diameter from 4 feet to 5 feet 6 inches, and have generally a cast-iron flat bottom with wooden sides. They ordinarily hold 1,200 to 1,300 lbs. of ore, and three is the usual number allowed per battery of five stamps, but sometimes two will be found sufficient. Each pair of pans requires one settler. In some districts copper plates are introduced into the pan, and much of the amalgam is found attaching to these, but the more usual system is to employ settlers entirely for the collection of the quicksilver and amalgam, after the pans are discharged. While the pulp is being worked in the pans, which usually takes 6 to 8 hours, steam is introduced to heat the mass and promote the chemical reactions; sometimes live steam is introduced direct from the boilers into the charge, but more commonly the pan is furnished with a false steam-bottom, and heated with the exhaust from the engine. The bottom of the pan is protected by cast-iron dies, and the muller is furnished with adjustable shoes, so that the wearing surfaces are renewable if it be necessary to grind the ore. The shoes and dies can be brought together when grinding by means of the hand-wheel, and

screws on top of the spindle, or the muller can be raised above the dies for circulation and mixing only.

The screw on the settler-driver, should, unlike the pans, be right-handed, for besides being more convenient in case of a belt slipping, the power applied to turn the screw, helps the muller to revolve. A settler should never be allowed to foul by an accumulation of heavy matter at the bottom, it is a positive preventive of good work. It is, however, easier to advocate this than to do it. An apparently natural remedy, viz., a liberal use of water, tends rather to aggravate the difficulty ; there is a point in the thinning when the quicksilver will be precipitated, but the heavy sand be held in suspension.

If, after the charge is run out (which should leave about 8 inches of pulp in the settler), a pan is drawn and no water is added for half an hour, the warm charge will gather and carry the heavy sand ; now enough water only is added to reduce it to the appearance of still some thickness, and this is all the water that is used in the charge. A horn spoon will show its success in advance of results.

Settlers are generally made with wooden sides 8 feet in diameter inside the staves, an automatic syphon-tap being provided for the discharge of the quicksilver and amalgam. Around the bottom a groove is cut, starting from nothing on one side, and gradually deepening to the syphon-tap opposite, in which all the quicksilver is carried to the outlet.

The muller-plate attached to the driver-arms, is shod with wooden plough-shoes, which are sometimes, however, attached direct to the arms themselves. The speed of the settler is generally about 15 revolutions per minute. Concentrators for a silver-mill must of necessity be simple and capacious. Good agitators (shovelled out often) are profitable. In some cases sand sluices are very effective, consisting of a broad sluice 20 to 24 inches wide, in which at intervals of 8 or 10 feet vertical strips are fixed at the side to hold movable riffles. The riffles (battens of wood) are laid in, and the sands run over them for a time (say one or two hours), when another course of riffles $\frac{1}{2}$ inch thick or less is laid on the first ones.

This is repeated until the sluice is full, when it is shovelled out, meantime allowing the sands to run through a duplicate sluice, at the side. Such sluices should have a grade of about $3\frac{1}{2}$ inches per rod to keep them under control, as by starting with a thin riffle at the bottom a strong current may be produced, whereas the introduction of a thick riffle will give a deadened current.

These sluices are an advantage where blankets are profitable, and if followed by blankets relieve the latter from much coarse, heavy material.

A blanket sluice should have a grade of about 3 inches in 7 feet. The stock of mercury in a silver-mill should be large to begin with. In a dry-crushing silver-mill the loss is usually from $\frac{1}{2}$ to $\frac{3}{4}$ lb. per ton of ore. A 10 stamp mill usually requires from 200 to 250 lbs. of mercury per month to make up the loss. The quantity needed in stock depends on the richness of the ore, but is approximately 1,500 lbs. in the pans, 1,500 lbs. in the settlers and in circulation, and 1,500 lbs. locked up in amalgam, so that a total stock of 2 to 3 tons would be necessary for starting with.

Roasting Milling.—The Reese River Process.

After passing the rock-breaker, the ore is dried by passing through a continuous revolving-drier or shelf dry-kiln beneath the breaker, the dried ore being taken by car or else run through shoots (lined with sheet iron and regulated by gates) to the automatic feeders, if the fall admits of it. The stamps are fed while the ore is still hot, the pulverized product being conveyed to the elevator, by which it is carried to the iron storage-hopper of the roasting-furnace. In the furnace the ore, with the addition of common salt, is desulphurized and chloridized, thus preparing it for the pans and settlers. After roasting, the ore is spread on a cooling-floor, and is taken in cars as required to the pans. Amalgamation follows on the same plan as in wet crushing-mills. In old type mills it was formerly the practice to employ drying-floors of boiler or cast-iron, from which the ore was shovelled to the stamps, in place of the more modern arrangement of an automatic revolving-drier.

The Boss Process.

This process marks a new epoch in the milling of silver ores in the United States and in Mexico, as it presents claims to superiority in many respects over the old system of pan-amalgamation. The large saving in labour and fuel, increased cleanliness, reduced wear and tear, and other features that will be mentioned later on, combine to make it a favourite with mill-men. It does away with the large pulp settling-tanks and consequent shovelling and handling of the pulp, which is a serious item of cost in ordinary wet treatment; it saves the erection of the tanks and the space they occupy; and no slum-pump or agitators are required.

The buildings and cost of erection for a continuous mill are less expensive than for an ordinary one, as they require less grading and retaining-walls and cover less area. The ore passes through the grizzley and crusher in the usual manner and down the automatic feeders to the stamps. The pulp flows from the battery through pipes to the special

grinding-pans (the product of ten stamps passing through two in succession). The pulp is then conveyed by pipes to the first amalgamating-pan, and flows continuously through the lines of pans and settlers. The tailings are run off and led over concentrators. A special feature of this process is that the pulp in the amalgamating-pans is always kept thin, instead of being about the consistency of thick cream, as usual in the ordinary pan process. The quicksilver is charged to the pans by means of pipes from the distributing-tank, and the amalgam flows direct to the strainer. The chemicals are supplied to the pans by two chemical-feeders. Steam syphons are provided for cleaning out the pans, and for conveying the pulp past any pan when it is necessary to cut it out of the series for repairs. The main-line shafting runs directly under the pans and settlers, each of which is driven from it by a friction-clutch. This arrangement of separate clutches for each pan and settler is very convenient, as any number or any one pan and settler can be stopped in case of accident for cleaning out, without having to stop the whole line.

All the water from the batteries must pass through the pans, so that all the slimes are treated ; there is less loss of mercury and a true sample of the tailings can be obtained, a matter of much greater difficulty with the old method. Heating by exhaust steam is stated to be one of its economical features, obviating any strain upon the pan, which is heated indirectly through the hollow steam-bottom. Where changes were made from live to exhaust steam in some mills it is said to have saved as much as £2 10s. 0d. to £3 2s. 6d. per day for cord wood.* By using special grinding-pans, the ore can be crushed through a coarser screen in the battery, and the finer grinding can be afterwards accomplished in the pans, thus obtaining increased capacity.

Though it does not pretend to cope with rebellious ores which are unsuited to such treatment, the Boss process is without doubt a great improvement over ordinary pan-amalgamation in tanks. If the latter process be employed, the tanks are filled in succession ; the pulp being conveyed to them through a launder, by means of which the supply is cut off as each vat becomes full. Arrangements should be made to settle as much of the mineral as possible by allowing the water to circulate through the empty tanks before passing to the slum-pit outside, the escape or tailings water being turned into each tank after emptying it of sand. Each tank in turn thus receives the water after passing through the other tanks, and becomes the final one of the series. In some mills a certain number of tanks are kept employed in settling the sands, while the

* Messrs. Fraser-Chalmers, *Catalogue, No. 4*, page 82.

remainder are used up for the slimes; in others the capacity of the vats is large enough to settle the sands and slimes together. Before charging the pulp into the pans it is usually shovelled into heaps on the platform in front of the pans, which is slightly inclined towards the tanks to drain the water back into them. In the pan-treatment a proper consistency of the pulp, a proper degree of heat, and clean quicksilver, are matters of the chiefest importance.

While charging the pan the muller for grinding is kept revolving and lowered, water having been previously run in so as to fill the pan to within 12 to 18 inches of the edge, and heated with steam. Some mill-men favour direct heating with live steam, others by means of a jacket or false bottom. The charge must be heated nearly to boiling-point by turning on steam again during the grinding. At the commencement of the grinding the pulp is thin, but after a couple of hours it will acquire the proper consistency for receiving the quicksilver, which becomes diffused (by the heat and grinding) in small globules through the mass. The pulp should be thick enough to cling to a wooden paddle dipped in to test it, showing particles of mercury evenly disseminated through it, so that the charge will carry the quicksilver in suspension. The salt (say about 10 lbs.) is added as soon as the pan is charged, and 2 lbs. of sulphate of copper (or whatever proportion is used), half an hour later. After the pulp is heated to about 180 degs. Fahr. steam is cut off, and the muller should be lowered gradually during the progress of grinding. When finished (after about 2 hours), some 200 lbs. of mercury are added to a 1,200 lbs. charge of pulp. The grinding is then sometimes continued for another half-hour or an hour, when the muller is raised and the pan run with the muller up, for 3 hours more. There is, however, less chance of flowering if the bulk of the mercury is not added until the grinding is entirely finished. A quarter of an hour before drawing the charge, sufficient water is added to fill up the pan, thinning the pulp thoroughly, so that it will flow readily out of the pan, and cooling it.

The work should be so arranged as to charge and discharge a pan every 6 hours, which gives it a capacity of about 2½ tons. The pans should be discharged in succession, not all simultaneously; that is to say, as soon as one or two pans have been discharged and refilled, after a certain interval (depending on the number of pans in the mill) the next pan or two should have completed their 6 hours' work, and be ready to undergo the same process.

When discharged, a stream of water should be directed into the pans,

to rinse them out thoroughly. Everything thus flows to the settlers, and through the partial dilution of the pulp, the quicksilver settles to the bottom, and is collected in the syphon. During the discharge of the pans the settler arms are kept revolving, and after a short interval a spray of water is turned on, and allowed to run till the settler is full. It is then turned off and the muller is allowed to revolve for an hour. This allows the quicksilver to collect and settle. An abundant stream of cold water is then let in and the settler is allowed to discharge through the different plug-holes, commencing with the top one, the operation being timed so that the bottom hole is reached just in time to receive the next charge. Once a week or oftener the settlers should be cleaned out, and the coarse sand and sulphides accumulating in the bottom are re-worked in the pans. Generally two settlers discharge into one agitator, and a constant stream of water should run into them. They collect some coarse sand containing a little quicksilver, amalgam, sulphides, and a quantity of iron which is worked-up in the clean-up pan. The floors should be kept as clean and free from dirt as possible. All drains should lead into the agitators, and unless the weather is too cold the quicksilver floor should be sluiced down with a hose daily.

Retorting Amalgam.

The retorts for retorting silver bullion are generally cylindrical or square with the corners rounded off, and containing shelves for several iron dishes. They should be heated to a bright cherry-red heat before commencing the retorting, otherwise it is difficult to drive off the last traces of quicksilver. A serious loss is entailed by a retort bursting, not an uncommon occurrence even with the greatest care. They must therefore not be fired too strongly, and must be strongly made and well braced. The mercury fumes are condensed by condensers acting on the Liebig principle, the quicksilver being caught in a bucket of water, into which the end of the pipe from the retort dips; care being taken that the water is not able to run up into the retort as it cools by the end of the pipe being too deep under water. After the retorting is finished it is advisable to leave the retorts to stand for several hours before withdrawing the bullion.

For cleaning quicksilver from impurities, which become mechanically mixed with it, the quicksilver strainer invented by Mr. H. H. Oakes is recommended by Mr. Eissler and is described by him in detail.*

* *Metallurgy of Silver*, page 159.

General Remarks, Ores, etc.

The ores of silver which can be successfully treated by the Washoe process are those in which the metal occurs in a condition which will be acted on by quicksilver, assisted by heat, agitation, and certain chemicals (chiefly salt and sulphate of copper), which produce a certain quantity of sub-chloride of copper, through the secondary action of the metallic iron present. The chloride and sub-chloride of copper (both of which are liable to be formed), tend to reduce any sulphides of silver present, by exercising a chloridizing influence upon them, and at the same time decompose the sulphides of lead and zinc. The sulphate of copper, moreover, enhances the amalgamating energy of the mercury by tending to expel the lead, and by causing the formation of a small quantity of copper amalgam.

A list of the chief ores and minerals containing silver would comprise the following :—

Name.	Composition.	Per Cent. of Silver when Pure.
Naumannite	Ag_2Se	73·2
Eukairite	$Cu_2Se + Ag_2Se$	43·1
Hessite	Ag_2Te	62·8
Petzite	$(Au\ Ag)_2\ Te$	41·8
Sylvanite	$(Au\ Ag)\ Te_3$	10 to 15
Argentite (silver-glance)	Ag_2S	87·1
Stromeyerite	$Ag_2S + Cu_2S$	53·1
Sternbergite	$AgFe_2S_2$	34·2
Miargyrite	$Ag_2S + Sb_2S_3$	36·7
Pyrargyrite	$3Ag_2S + Sb_2S_3$	59·8
Proustite	$3Ag_2S + As_2S_3$	65·4
Stephanite	$5Ag_2S + Sb_2S_3$	68·5
Bournonite	$PbS + Ag_2S + Sb_2S_3$	26·1
Polybasite	$9\ (Ag_2Cu)\ S + (Sb\ As)_2S_3$	68·0
Tetrahedrite (Fahlerz)	$(CuAg)_2S + (SbAsBi)_2S_3 + (Fe\ ZnHg)S$	variable.
Xanthoconite	$(3Ag_2S, As_2S_5) + 2\ (3Ag_2S, As_2S_3)$	64·00
Fire blende	$AgSbS$	62·3
Freieslebenite	$Pb_2Ag_3Sb_3S_8$	23·8
Cerargyrite (horn-silver)	$AgCl$	75·33
Bromyrite	$AgBr$	57·40
Embolite	$Ag\ (Cl\ Br)$	61 to 71
Iodyrite	AgI	46
Native silver		100·00
Arquerite (native amalgam)	$AgHg$	34·8
Electrum (native alloy of gold and silver)		27 to 32·7

Minerals, etc., often containing silver in small quantity :—

Galena	PbS
Blende	ZnS
Pyrite	FeS_2
Chalcopyrite	$CuFeS_2$
Erubescite	Cu_3FeS_3
Mispickel	$FeS_2 + FeAs_2$
Altaite	PbTe
Clausthalite	PbSe
Nagyagite	(Pb AuAg) $(Te, S)_2$
Chivialite	$(Cu_2Pb)\ S + \frac{1}{2}Bi_2S_3$
Dufrenoysite	$PbS + As_2S_3$
Enargite	$3Cu_2S + As_2S_3$
Cupel bottoms, dross litharge sweepings, etc.	
Slags, etc.	

The presence of sulphides of iron, copper, lead, zinc, and antimony, interferes with the success of the amalgamation process in several ways. They foul the amalgam and check the reactions of the chemicals on the free-milling minerals, and carry off in their refractory combinations a portion of the silver which the latter contain. It often happens that while the upper decomposed surface-ores of a vein are free-milling, as depth is attained (beyond the decomposing action of the air and surface waters) they change in character, through sulphides and base metals making their appearance in the ore.

Occasional natural deposits of chloride of silver and some rare instances of native silver, unaccompanied by sulphides, form, with certain decomposed ores, the chief types adapted to free-milling; though the process being cheaper than roasting-milling, ores are sometimes worked by it, which should properly be roasted, but the percentage saved in such cases is correspondingly low. Ores carrying quite a large percentage of base minerals may be worked by roasting-milling, but in many cases it is more profitable (when the conditions admit of it) to treat such ores by concentration and smelting. The presence of certain minerals in combination may render the chloridizing roasting of an ore extremely difficult. The Silver King mine in Arizona may be cited as an instance of this, as mentioned by Mr. Aaron.*

The ore in question consisted largely at one time of fahlerz, chlorides, bromides, and oxides, in a gangue of quartz and heavy spar, and being of high grade it proved well adapted to treatment by the Kiss lixiviation process, for which it had to undergo a preliminary chloridizing roasting.

It was found that this ore sustained a serious loss of silver by volati-

* Report of the Director of the United States Mint.

lization during roasting; an extra draught produced by opening both ends of the fireplace somewhat mitigated the difficulty, but it was only finally overcome by introducing steam into the furnace (as originally suggested by Dr. Percy) which successfully met the difficulty; the volatile metal chlorides (to which the volatilization of the silver is mainly owing) being decomposed and converted into oxides, with the instantaneous production of hydrochloric acid. Unfortunately, however, as depth was reached in the mine the character of the ore changed. The proportion of chloride and tetrahedrite fell off, and zinc blende and galena became more abundant, and this led to far more serious difficulties.

The roasting became slow and tedious; while previously a charge of 5 tons could be well roasted in 14 to 16 hours, converting about 95 per cent. of the contained silver into chloride. The percentage of soluble silver in the roasted ore decreased also somewhat, causing richer tailings, and as the grade of the ore likewise fell off, a serious diminution in the output of bullion ensued. The ore, moreover, developed a tendency to ball and form crusts on the furnace walls. The balls were peculiar, being perfectly spherical and of all sizes from a pin's head to a marble, extremely hard, and consisting of concentric layers. The ore being crushed wet and received into settling-pits, no doubt operated disadvantageously in this instance.

When the ore was by no means at its worst, analysis showed it to contain 12 per cent. of zinc—equivalent to about 18 per cent. of blende, 6 per cent. of lead as galena, a good deal of antimony, a little arsenic, a very little iron and copper, and trifling quantities of cadmium, selenium, tellurium, and bismuth. The conjunction of antimonial and plumbiferous minerals with zinc blende tends in fact to make roasting difficult. The character of the gangue also exercises a great influence on the roasting. The presence of quartz is advantageous, spar or gypsum is not troublesome, but earthy carbonates are detrimental, and magnesia bad.

At one time the ore contained less quartz and spar than formerly, and more of the so-called porphyry of the district, which contains magnesia in abundance.

In the case of ore which balls in the furnace when roasted with salt, the usual practice is to roast without salt, to a certain stage when the salt is added and the heat increased; but the presence of metallic silver and the absence of a fair proportion of iron rendered this method inapplicable.

The next idea tried, that of roasting to complete oxidation without salt, and then chloridizing by an addition of calcined copperas and salt, has been used with a slight modification on some of the worst ores with good results.

Another successful plan was to mix a certain proportion of sand, about 7 per cent., with the charge. The sand used contained a little silver, being the coarser portion of a pile of rather rich tailings from previous concentration. The addition of one-third of clean quartzose silver-ore was found to act favourably, 95 per cent. of the chloride being got out in 24 hours with 3 ton charges.

Mr. Stetefeldt has lately introduced the plan of drying and roasting ore with gas, at the Holden mill, Aspen, Colorado, and at the Marcac mill, Park City, Utah, where lixiviation is employed for the treatment of the ore. The former plant was put in operation in November, 1891, and consists of four double shelf-driers, with one 6 feet diameter Taylor revolving-bottom producer, and one large Stetefeldt furnace, with a Taylor producer, also 7 feet in diameter. Mr. Morse, the general manager of the Holden works, states that, on a recent run of 4,631 tons of ore, 96·4 lbs. of coal were used per ton of ore roasted, costing 14·45 cents. The coal, consisting of a mixture of about equal proportions of Colorado Newcastle and Sunshine coal, costing 3·00 dols. per ton delivered at the mill. The composition of these coals are :—

		Fixed Carbon. Per Cent.	Volatile Matter. Per Cent.	Ash. Per Cent.	Water. Per Cent.
Colorado Newcastle	I.	55·9	35·9	5·4	—
,, ,, ...	II.	48·6	37·95	11·6	1·7
,, Sunshine	III.	48·0	43·0	7·5	—
,, ,, ...	IV.	37·1	36·3	23·8	2·8

This, Mr. Stetefeldt states, is the cheapest drying and roasting on record in any silver-mill, and the introduction of the system of employing gas producers is making rapid progress in silver-milling. Full details of the cost of drying and roasting at Aspen on a run of 12,000 tons of ore are given in the *The Engineering and Mining Journal*, New York, of June 25th, 1892.*

When ores contain a very large percentage of sulphur : *i.e.*, are exceptionally heavy. Mechanical roasting furnaces cannot, as a rule, compete with the old-fashioned reverberatory furnace, but, with a moderate amount of sulphur, many of them give excellent results. The cost of roasting (chloridizing) in the Stetefeldt furnace is said to vary from 16s. to £1 0s. 10d., and the furnace is said to cost £625 to build. A Brückner furnace, which is a favourite in small mills, will treat from 3 to 4 tons (in exceptional cases 10 tons) in 24 hours. The cost of roasting in it varies from 10s. to £1 per ton. An improved form, with double cylinders set tandem, is estimated to roast 20 to 40 tons of refractory ore (in inverse proportion to the percentage of sulphur it carries) at a cost

* Page 660.

of £4 3s. 4d. per day, *i.e.*, 2s. 1d. to 4s. 2d. per ton. Mr. Brückner's estimate of the cost of a double-cylinder plant is £2,255 erected.

In many of the best modern mills where dry-crushing is practised the furnaces are placed at the extremity of the battery line, a little behind it, and not in front, as they used to be.

Mechanical furnaces of the improved Brückner and Howell-White type (a modification of a Hocking and Oxlands calciner), and the Stetefeldt and O'Harra are amongst those in most general use.

If the ore requires roasting, dry stamps are invariably used, the mortars being covered in with a wooden housing, to which exhaust fans are attached to draw off the dust into pockets (emptied at intervals), and the dry ore is moved by screw-conveyors or horizontal endless-belt tables to the feed-pocket of an elevator, which raises it to the hopper of the furnace if a mechanical roaster be employed.* When the ore is roasted a chlorination assay† must be made of every charge.

The floors of modern wet-crushing mills are laid slightly inclined towards a reservoir connected with the pulp-tanks, double-planked and tarred, and the mill supply of quicksilver is almost always handled mechanically by a mercury pump. It is imperative, however, that it should be of first-class make, as a poor device for handling quicksilver is more extravagant than hand labour.

The variations in the details of the plant and method of manipulation are capable of so great a number of permutations that it would be useless to attempt to go into the subject fully in this paper. The author should, however, state that the fineness to which the ore can be reduced is to no small extent determined by the capacity of the settlers to work off the coarse sands without loss of mercury.

Although it has been found by experience that some ores roast as well if crushed through a 30 as they would through a 40 mesh screen; some heavy ores, *i.e.*, those that contain a great deal of sulphur, give a low chlorination and extraction, unless crushed finer, say to a 50 mesh. The limit of coarseness to which it is ordinarily practicable to carry crushing with stamps is, the author believes, about 30 mesh.

* A new form of elevator and conveyor made by the Jeffrey Manufacturing Co., of Columbus, Ohio (a description of which is given in *The Engineering and Mining Journal*, New York, of March 4th, 1893, page 201), consisting of a steel cable, to which a number of iron diaphragms of suitable shape are attached at intervals, which travel in a trough of corresponding shape, would seem to be particularly applicable to this purpose.

† The method of doing this is excellently described by Kustel, *Roasting of Gold and Silver Ores*, second edition, 1880, page 32, *et seq.*

It cannot be too strongly emphasized that one of the most important points in pan-amalgamation is cleanliness about the works, and the use of clean quicksilver; though order and neatness, with the polish that comes of the use of elbow-grease, are factors of economy that ought to be naturally looked for in all mining plant.

Bichloride of copper is supposed to be the active agent in the Washoe process in the reduction of sulphide of silver; just as bichloride of mercury* attacks gold and amalgamates with it when ordinary quicksilver will not touch the yellow metal.

If it is worth while putting up a well-constructed mill building (which cannot be done without corresponding expense) it is worth while keeping it in first-class repair.

The cost of an ordinary 20 stamp dry-crushing plant, including rock-breaker, mechanical-drier and roasting-furnace, cooling-floors and elevators, with stamps and conveyors, and the necessary pans, settlers, sluices, etc., with a quicksilver-tank system, will not, in most cases, fall far short of £7,100, in London, exclusive of local freight and erection charges. It would weigh about 234 tons.

The cost of a 20 stamp wet-crushing silver-mill plant, with rock-breaker, automatic-feeders, stamps, pulp-tanks and pans, settlers, etc., will not generally come to less than £6,525, and weigh about 186 tons.

The cost of treatment, milling ores wet, varies from 12s. 6d. to £1 17s. 6d., employing the Washoe process. A high average being about 18s. 9d. per ton.

The cost of treatment, milling ores dry†, varies from £1 5s. to £2 11s., and when the ore requires roasting it will average from £1 13s. 4d to £3 2s. 6d., and sometimes as high as £5 16s. 8d. per ton.

The great variations in silver ores, conditions of working, and methods of extraction make it impossible to give more than very general estimates of cost, as it fluctuates frequently in the same district, with differences in price of fuel, labour, freight, chemicals, etc.

* The Designolle process.

† The writer alludes here to a modification of the Reese River process, which dispenses with roasting; as examples of which, we have the Eberhardt at White Pine, and the Lancaster mill at Tuscarora. It is practised on the grounds that a much higher percentage is saved, if the ore contain chlorides, as the finely divided horn-silver is likely to be lost if crushed wet, although the slimes are thin. It is applicable also to some ores which produce an excessive amount of slime, which escapes the settling-tanks.

A 25 stamp wet-crushing silver mill, running 24 hours, generally needs the following crew of men :—

2 rock-breakermen.
2 battery feeders.
2 amalgamators.
2 engineers (and in some cases 2 firemen).
1 mechanic.
1 foreman and assayer.
2 assistant amalgamators.
3 tank-men.

The Grand Prize* (a 20 stamp dry-crushing and chloridizing mill) employs :—

No. of Men.		Wages per 12 Hours' Shift. £ s. d.
2	amalgamators	at 1 0 10
2	„ helpers	„ 0 16 8
2	chloridizers	„ 1 0 10
2	„ helpers	„ 0 16 8
2	battery feeders (tenders)	„ 1 0 10
2	engineers (drivers)	„ 1 0 10
4	firemen	„ 0 16 8
1	melter and retorter	„ 0 16 8
6	dry-kilnmen	„ 0 16 8
1	blacksmith	„ 1 0 10
4	labourers	„ 0 16 8
28		

The Lancaster* (a 10 stamp dry-crushing raw-amalgamating mill) employs :—

No. of Men.		Wages per 12 Hours Shift. £ s. d.
2	amalgamators	at 1 0 10
2	„ helpers	„ 0 16 8
2	battery tenders	„ 0 16 8
2	engine drivers	„ 1 0 10
3	firemen	„ 0 16 8
2	dry-kilnmen	„ 0 16 8
3	labourers	„ 0 16 8
16		

Examples of the Washoe Process.

The ores of Mineral Hill, Nevada, consist of chloride of silver, bromide of silver, argentite, polybasite, stephanite, carbonate and molybdate of lead, carbonate of copper, and some manganese, occurring in a limestone-formation in irregular deposits. Mr. Eissler† states that, being of a complex character, they were originally treated by roasting milling, but he

* Egleston, *Metallurgy of Silver*, page 437. † *The Metallurgy of Silver*, page 155.

subsequently worked them successfully on the Washoe principle by dry-crushing and amalgamation (the modification of the Reese River process, before alluded to).

A point of special interest is the presence of bin-oxide of manganese in the ore. This mineral appears to have a deleterious effect on the amalgamation, its presence being indicated in the settlers by a thick froth, which in spite of dilution with water, carried off flowered quicksilver. The charge, Mr. Eissler states, which gave the best results, was found to be 1,500 lbs. of ore mixed in the pans with 15 to 20 lbs. of salt and 3 to 5 lbs. of bluestone. Treating 18 tons per diem, the cost was as follows:—

	£	s.	d.
Superintendent, who also acted as assayer	2	1	8
Master mechanic	1	5	0
Carpenter	1	5	0
Two engineers at £1 0s. 10d., 1 day and 1 night ...	2	1	8
Two men tending rock-breaker at 16s. 8d.	1	13	4
One man at dry-kiln, and to take battery samples, 1 day and 1 night	1	13	4
Two battery-feeders, 1 day and 1 night at 18s. 9d. ...	1	17	6
Two pan-men during the day	1	13	4
Two pan-men and one retorter at night...	2	10	0
400 lbs. of salt at 3d.	5	0	0
8 cords of wood at £1 5s.	10	0	0
Loss of mercury, 30 lbs. at 5s.	7	10	0
Wear and tear of iron, and repairs	4	3	4
Oil and incidentals, sulphate of copper and assay materials	3	2	6
	£45	16	8

Cost per ton, £2 10s. 11d.

The ores of Pioche, Lincoln County, Nevada, which contain on the average 3 to 5 per cent. of lead (cerussite and galena) have been treated successfully by the ordinary Washoe process, the ore being worked up to over 82 per cent.; when assaying £27 1s. 8d. per ton and yielding bullion, the average fineness of which was somewhat below 700, containing lead and some copper.

To extract the greater part of the lead, the quicksilver and amalgam after leaving the settlers was strained in sacks suspended in a large box filled with water, heated with steam by a ½ inch pipe. Lead amalgam at the temperature of boiling water remains liquid, and will therefore strain through with the excess of quicksilver. As a certain amount of silver and copper amalgam also passes through, the mercury is run into a smaller box cooled with water, and when cold strained in the usual way,

leaving an amalgam of lead containing a small amount of the other metals. This lead amalgam, when retorted, gave bullion containing 6 to 20 per cent. of silver, very little copper, and only a trace of gold. The amalgam in the first sacks gave bullion from 550 to 680 fine in silver, and finer in inverse proportion to the amount of copper in the ore.

The amalgam from the second straining of the quicksilver, with ore of normal character, gave bullion 60 to 200 fine. When the ore was amalgamated without chemicals the bullion was 300 to 350 fine, and when amalgamated with salt and bluestone, but not strained in hot water, 400 to 450 fine. The ore contains on an average £1 0s. 10d. in gold to over £20 16s. 8d. worth of silver, this proportion remaining very constant. 45 to 55 per cent. of this gold is extracted. The bullion contains 0·0003 to 0·0015 parts of gold, and occasionally as high as 0·0030 parts. With bullion 500 to 600 fine (after passing through the hot-water straining process) the loss of quicksilver amounted to $4\frac{3}{4}$ lbs. per ton of ore, due probably to the formation of chloride of lead and the subsequent formation of sub-chloride of mercury. Another source of loss was the formation of floured lead-amalgam, which had the dull appearance of lead and floated off in flakes on the surface of the water. The chloride of copper formed rapidly destroyed the castings of the pans. The proportion of retorted bullion to amalgam in working the Comstock, White Pine, and Idaho ores is as 1 to $5\frac{1}{2}$ to 6; in amalgam containing a large amount of copper as 1 to 7 to $7\frac{1}{2}$; and in very base amalgam as 1 to 4.*

The ores of the Silver Reef in the Harrisburg district, Utah, are of a remarkable character, consisting of chocolate-coloured sandstone with fine chloride of silver disseminated through its mass, and where organic remains (such as leaves and stems of trees) are found embedded in it, the silver is present in a pure metallic state. In the Stormont mine, the ore is found in a zone 10 to 100 feet thick, often associated with fossil-remains, etc., and is bounded by red sandstones above and white sandstone below. The ore is very easily crushed and disintegrated, a 750 lbs. stamp putting through 7 to 8 tons per 24 hours' crushing through a 40 mesh screen, so that 5 to 10 stamps will more than supply 12 pans, working $1\frac{1}{2}$ tons pan charges. Considering the size of the mills, the cost of milling is in consequence extremely low. The ore averages £6 5s. per ton. The tailings vary greatly in richness, according to the character of the ore. With sandstone ore, they will often carry 12s. 6d. per ton, while with shaly ore they may run £2 1s. 8d. or more.

* Eissler, *Metallurgy of Silver*, page 133.

Mr. R. P. Rothwell gives the cost of a year's working at three of the mills of the district as follows:—

Per ton of 2,000 lbs.	Christy M. and Mg. Co. 14,249 tons. £ s. d.	Stormont Co. 9,983 tons. £ s. d.	Leeds Co. In 1878 12,064 tons. £ s. d.	Leeds Co. In 1879 4,679 tons. £ s. d.
Labour & salaries	0 11 10½	0 12 4½	0 9 2	—
Bluestone .. (2·1 lbs.)	0 1 3½	(1·¾ lbs.) 0 1 1		
Mercury ... (1·22 lbs.)	0 2 5	(1·13 lbs.) 0 2 4½		
Salt .. ... (25·8 lbs.)	0 2 1½	(20·00 lbs.) 0 1 2¾	0 13 5	—
Fuel	0 5 5½	0 1 10¾		
General supplies	0 3 7½	0 1 10½		
Incidentals ...	0 1 8½	0 0 6		
	£1 8 6	£1 1 4	£1 2 7	£0 17 2
Hauling ore to mill	0 3 0½	0 8 4	0 1 4	0 1 0½

In parts of Mexico, silver ore which falls below £6 5s. per ton is not available for the Washoe or patio processes, owing to the excessive cost of transport, and fuel. At Guanajuato, for example, packing on mules costs 14s. 7d., and treatment of the ore £2 7s. 11d., whilst mining, pumping, hoisting, etc., add a further charge of £2 10s. Wood costs £2 1s. 8d. and coal £4 11s. 8d. per ton. The district, however, is said to be a rich one, one group of mines north of the city of Guanajuato being credited with a production of 312,860,000 dollars between 1548 and 1889, out of a total of 650,000,000 dollars worth of silver obtained from the district.

Mr. W. L. Austin gives* the subjoined particulars of the cost of running a wet-crushing silver-mill in the Tombstone district, stating however that, owing to the arrangement of the plant, which works under some disadvantages consequent upon its alteration from dry-chloridizing to wet-crushing and other reasons, a reduction of 20 per cent. in the cost of milling could be effected under more favourable circumstances.

A novel feature to be noticed is that the shoots leading from the ore-breaker to the bins are not only provided with the ordinary grizzleys, the bars of which are set ⅜ inch apart, but the bottom of the shoot itself is fitted with a shaking-frame covered with screens of the same mesh as those used in the battery.

This relieves the batteries materially, and decreases the amount of slimes, increasing the capacity of the mill 5 per cent. or more, depending on the fineness of the ore and its percentage of moisture, as the ore fine enough to pass the screens, goes direct to the pans without passing through the battery at all.

* "Silver Mining in Arizona." *Trans. Am. Inst. Min. Eng.*, vol. xi., page 91.

The average gross weight of the stamps is between 750 and 800 lbs. The shoes, weighing 120 lbs., have an average life of one month, when worn down to about 35 lbs.

Hendy Challenge feeders are used, and the stamps crushed 2·9 tons of medium hard rock in 24 hours for the first six months they were in operation, using a 30 mesh screen.

After 4 hours' grinding in the pans, the average extraction was found to be about 81·04 per cent., and nothing material was gained by prolonging the treatment, but by the use of salt and bluestone 87 per cent. could be got out of the ore, which contained only 7 per cent. of its silver as chloride.

So long, however, as coarse crushing was adhered to in the battery, the bullion was much debased. The remedy was found to be to crush finer, using 35 mesh screens and amalgamating without grinding in the pans, by which means the bullion was kept at ·970 fine without sacrificing the milling percentage. The ore contains about 8s. 4d. in gold, and 43 per cent. of this was also recovered. It may be mentioned that the debasing of the bullion was chiefly due to lead, which lowered it down to ·200 and ·300, and whenever wulfenite appeared in the ore this was remarked to a much greater extent than when the lead was in the form of cerussite or galena.

The average cost of milling for five months was with 20 stamps :—

	£	s.	d.
Labour	0	10	6
Fuel...	0	4	4½
Chemicals and mercury	0	3	2½
Lubrication	0	0	2
Illumination	0	0	1½
Castings	0	1	4½
Supplies	0	0	8
Per ton	£1	0	5

The cost of labour (subdivided) treating 1,730 tons, based on one month's work was estimated to be :—

	s.	d.
Crushing	2	2
Amalgamating	2	9½
Power-pumps, etc.	1	11½
Foremen, etc.	3	7½
Tailings-pits	0	5½
Per ton	11	0

The loss of mercury averaged about 5s. 5d. per ton, and about 0·11 cords of wood,* and 1,200 gallons of water, were consumed per ton milled.

Mr. Austin also gives another instance of the cost of wet-crushing in Arizona, estimated on 2,643 tons crushed in a 20-stamp battery, furnished with a rock-breaker and automatic feeders, separating the coarse sands in hydraulic sizing-tanks, and working each sand-class by itself, for which the ore alluded to was particularly suitable, being entirely free from base metals with a gangue of light specific gravity.

The stamps weighed 800 lbs., and put through on an average 5 tons per head in 24 hours.

Cyanide and lime, in the proportion of 14 lbs. of the former chemical and 120 lbs. of the latter per 100 tons of ore, were used in the pan treatment. The mercury was pumped through the mill for distribution.

The motive power was a 200 horse-power horizontal engine (42 inches by 20 inches cylinders, run at 60 revolutions per minute), and the boilers, which were tubular, 15 feet 6 inches by 54 inches, carrying 85 lbs. steam pressure, burnt 16 cords of wood irrespective of the boiler for the pump, which burnt 8 cords per week. The cost per ton of ore was :—

	£	s.	d.
Labour	0	5	1½
Supplies	0	7	7
Assaying	0	4	5½
	0	17	2

The cost for labour as given above is thus subdivided :—

	£	s.	d.
Crushing	0	1	1
Amalgamation	0	0	10
Power, pumps, and repairs	0	1	8
Foreman, melter, etc.	0	1	6½
	0	5	1½

The cost of material as given above is thus subdivided :—

	£	s.	d.
Mercury	0	1	9
Chemicals	0	0	3½
Castings	0	1	2½
Illumination and lubrication	0	0	3½
Fuel, including pump	0	3	3
Supplies	0	0	9½
	0	7	7

* This includes pumping the water supply 200 yards (with a lift of 100 feet vertical); 7 cords of mixed wood (black oak, white willow, and pine), costing £1 17s. 6d. per cord, were used on the average per day. The mill engine was provided with a Meyer cut-off.

The consumption of wood per ton of ore was 0·15 cords, and of mercury 0·96 lbs.

The bullion averaged ·995 fine.

Stated in gallons, the quantity of water used per boiler in a silver-mill is 7½ gallons per horse-power per hour. For each stamp, 72 gallons per hour; for each pan, 120 gallons per hour; and for each settler, 60 gallons per hour.

The consumption of wood (not including that used for roasting) in a dry-crushing mill is put by Prof. Egleston at about ⅓ cord per ton.

The power required for a Brückner roasting-cylinder is estimated at about 2 horse-power, and for the Howell-White 1½ horse-power.

The ore according to its baseness loses 3 to 15 per cent. by weight in roasting.

The value of stock on hand necessary to run a dry or wet-crushing silver-mill, including wood, mercury, castings, chemicals, etc., is very variable.

Examples of Roasting Milling Treatment.

The ore of the Ontario mine is a very base one, being composed of zinc, lead, and silver sulphides and silver chloride in a quartz gangue. This becomes baser with the increasing depth of the workings. The bullion obtained runs about 600 fine and contains no gold. The average grade of the ore treated is 100 to 130 dollars per ton, and the amount treated varies from 50 to 55 tons per day, though formerly (when it was less base) 65 tons were handled daily. The ore is roasted in Stetefeldt furnaces with the addition of about 15 per cent. (dry weight) of salt.

After leaving the furnace the ore goes to the cooling-floor, where it remains piled up for 18 hours. This increases the chlorination from 3 to 8 per cent. After being damped it is run to the pan-room. Each pan-charge of ore weighs 2,500 lbs., to which 1 per cent. of salt is added, and the pulp made up with the addition of hot water to proper consistency. The muller makes about 65 revolutions per minute, and is held 1 inch above the pan-bottom so that it does not grind. About 1 lb. of zinc and 300 lbs. of mercury are added to the pan after it has run 1 hour (and is hot), and amalgamation is continued for 7 hours. From the pans the pulp is drawn into settlers, which run 4 hours, making 40 revolutions per minute, and after running for 1 hour cold water is let in, and the overflow discharging the tailings is set running. The mill is worked up to from 88 to 92 per cent. of the value of the ore, that being counted as the amount chlorinated. The tailings carry off 8 to 12 per cent. of the silver.

The cost of treatment on a production of 50 tons per day was stated as follows* :—

No. of Men.	Occupation.		Per Day. Dollars.	Per Ton. £ s. d.
1	Foreman...	...	10·00	0 0 10
1	Assayer ...	...	6·00	0 0 6
3	Machinists	at $4·00	12·00	0 1 0
2	Carpenters	at 4·00	8·00	0 0 8
2	Blacksmiths	at 4·00	8·00	0 0 8
2	Engineers	at 4·00	8·00	0 0 8
2	Foremen	at 3·50	7·00	0 0 7
9	Dry-floormen	at 3·50	31·50	0 2 7½
3	Batterymen	at 4·00	12·00	0 1 0
6	Roasters	at 4·00	24·00	0 2 0
12	Cooling-floormen	at 4·00	48·00	0 4 0
4	Carmen	at 4·00	16·00	0 1 4
4	Amalgamators	at 4·50	18·00	0 1 6
1	Retorter	at 4·00	8·00	0 0 8
1	Melter	at 4·00		
4	Labourers	at 2·50	10·00	0 0 10
4	Watchmen	at 3·00	12·00	0 1 0
2	Ore-floormen	at 3·50	7·00	0 0 7½
3	Clerks	at 4·00	12·00	0 1 0
66			$257·50	£1 1 6

Supplies.		Per Day. Dollars.	Per Ton. £ s. d.
Salt, 10 tons	at $8·00	80·00	0 6 8
Quicksilver, 175 lbs.	at 0·50	87·50	0 7 3½
Wood, 15 cords	at 4·50	67·50	0 14 8½
Coal, 12 tons	at 8·25	99·00	
Castings			0 6 3
Oil and waste...			0 1 0½
Sundries, chemicals, etc.			0 2 1
Hauling from mine ...			0 2 0½
Charcoal, assaying, and melting			0 1 0½
			£2 1 1½

Total cost per ton, £3 2s. 7½d., exclusive of office expenses, general superintendence, repairs, and insurance.

The report of the Granite Mountain Company, Phillipsburg, Montana, one of the largest dry chloridizing roasting-crushing plants in America (running 153 stamps), gives the following particulars of working for the year ending July 31st, 1891 :—

Ore, tons crushed, wet,	72,623	= dry weight,	68,850
Salt, „ „	10,807	= „	10,645
Total	83,430		79,495

Average moisture in the ore, 5·2 per cent.; average moisture in salt, 1·5 per cent.

* R. P. Rothwell, *Trans. Am. Inst. of Min. Eng.*, vol. viii., page 557.

The salt and ore are mixed before crushing.

The ore averaged 50·59 ounces of silver, and the cost of milling was £2 1s. 8d. per dry ton, divided as follows :—

	£	s.	d.
Labour and superintendence ...	0	14	5
Salt	0	8	3¼
Fuel	0	6	10¼
Mercury	0	4	8
Castings	0	2	7½
Chemicals	0	1	2½
Water	0	0	6¾
Miscellaneous	0	3	0¾
	£2	1	8

Equal to £1 19s 5¼d. per wet ton.

The company's three mills of 20, 43, and 90 stamps respectively are known as A, B, and C, and have cost, with improvements up to date, £145,843 17s. 9½d., as shown by the trial balance sheet. The cost varied from £1 16s. 7½d. in mill C, which crushed 42,153* wet tons, to £2 11s. 10d. in mill A, which crushed 9,934† wet tons.

The saving of silver amounted to 90·7 per cent.

The report of the Elkhorn Company, of Montana, for 1891, affords another excellent illustration of close saving and business-like management.

The mill, which crushed 11,645 tons (dry) during the 12 months ending December 31st, 1891, saved 93·78 per cent‡ of the silver contents of the ore at a cost of (9·226 dollars)§ £1 18s. 5¼d., a close approximation to the figures just given.

Tailings Mills.

Silver-mill-tailings are generally concentrated on concentrators, or on blankets, in sand-sluices, and either leached, or treated in pans on the Comstock system (using chemicals) ; which has given rise to what are known as tailings, or auxiliary mills.

Tailings from silver mills can in fact often be treated profitably by storing them in dams, leaving time, air, and chemical agency to effect their oxidation, and then treating them raw in pans, no drying or chloridizing being required.

The chief drawbacks against this method are the uncertainty as to the time necessary for complete oxidation of the minerals present, which may be taken from 2 to 4 years, and the necessity of constructing large storage dams.

* Representing 40,067 dry tons. † Representing 9,318 dry tons.

‡ In 1890, 86·83 per cent. was saved, the increase in 1891 is attributed to using extra salt. § For analysis, see Appendix.

Mr. Eissler states* that, in some establishments of this kind in Nevada, the quantity of sulphate of copper supplied to the pans with this system of treatment varies from 3 to 6 lbs. per ton of tailings, while the salt amounts to 20 or 30 lbs. The pans are covered, and supplied with steam at a high temperature.

He places the yield at about 60 per cent., dealing with an ore worth £3 to £3 10s. per ton, and gives the current cost as follows :—

	£	s.	d.
Labour	0	6	0
Quicksilver lost	0	4	0
Salt	0	3	0
Sulphate of copper	0	2	6
Fuel	0	5	0
Castings	0	0	6
Total	£1	1	0

In Avery's tailings-mills in Washoe valley, where wood is £1 4s. per cord, the cost per ton is stated to have been 14s.

It is usual in Nevada for two sets of samples to be taken as a basis upon which to determine the value of the ore, when it is treated at customs establishments. One of these samples is taken from the waggon conveying the ore from the mine to the mill, the other from the mill after the ore has been crushed. The waggon sample is obtained by drawing from each waggonload of ore a sample of rock, and mixing the total number of samples of the loads sent to any one mill at the end of the day. This bulk sample is then quartered down, the mill being charged with the weight of ore, and the amount of gold and silver represented by assay. The mill sample, which is to serve as a check on the waggon sample, is taken by allowing the crushed ore as it comes from the battery to run into a pail, held at the end of the trough leading to the tanks.

A sample of this kind is taken in most mills every hour, in some every half-hour, and the accumulated samples are well mixed, dried, and reduced to a sufficient quantity for assay.

The pail or vessel in which the crushed ore is caught must not, of course, be allowed to overflow.

In some mills the samples are taken from the tanks after the sand has deposited itself in them, either from the surface or preferably with a tube sampler.

Strange as it may seem the waggon samples and mill samples generally differ in yield, and the former is usually the highest.

This was proved in the late celebrated suit tried before Judge Hebbard, Fox *versus* the Hale and Norcross Co. In practice, therefore, both assays are duly considered, and an adjustment arrived at.

* *The Metallurgy of Silver*, page 114.

A. D. Hodges, in an interesting paper on pan-amalgamation,* estimating on 46,500 tons of Comstock tailings treated at Drayton, Nevada, calculates the cost of treating them as follows :—

	s.	d.
Preparing and hauling	1	3½
Milling...	8	1½
General expenses	0	10¾
Total per ton ...	10	3¾

Some of the sand so treated seems to have run about 15s. 3½d. in silver, and 4s. 8½d. in gold, total £1, and yielded in bullion 75·8 per cent. of the silver, and 25 per cent. of the gold, total 63·8 per cent., whilst some of the slimes averaged £2 15s. 3d. silver, and 9s. 5½d. gold, total £3 4s. 8½d., and yielded 85·8 per cent. of the silver, and 42·5 per cent. of the gold, total 79·5 per cent.

The loss of quicksilver, treating sands at Drayton, ran under ½ lb., and with slimes from ¾ to 1 lb. per ton.

The bullion from the treatment of the sands did not run over 150 fine, and with the richer slimes the best results were got when it was from 200 to 250 fine.

In Dakota, it may be mentioned that with free-milling gold ores, it cost 4d. for mill labour, to produce 4s. 2d. worth of gold.

In Nevada, on the other hand, though over five times as much work is required per ton treated, it costs only about 3½d., but this seeming contradiction is explained by the higher grade of the Nevada ores.

A 20 pan mill, without any special arrangement for catching mercury working on tailings, made a gain of 700 lbs. of mercury in 6 weeks, which was but a small part of the mercury actually contained in the slimes.

In view of these losses it generally pays to employ concentrators to work up the mill-tailings. Sometimes a machine, known as a Varney amalgamator, is set in the tailings-sluice.

As battery slimes require no grinding it would seem most advantageous to charge the quicksilver direct into the pan, but experience has shown that better results are obtained by charging the chemicals with the slimes and thoroughly mixing them for about two hours with the muller up, and then adding the mercury, as the slimes form a pasty mass which might hold and carry off the finely divided quicksilver. In treating such material one-half its bulk in sand (tailings from the pans), and sometimes more, is added to the charge.

In many so-called free-milling ores of silver, or silver and gold combined, there are small quantities of sulphides of the base metals, not

* *Trans. Am. Inst. Min. Eng.*, vol. xix., page 231.

sufficient in quantity or value to make the ore suitable for roasting, yet enough to prevent a high extraction by free-milling, besides increasing the cost. Such ores can be best treated by the combined process of concentration and amalgamation.

Stamping the ore wet, passing it over copper plates, concentrating in vanners, and then amalgamating the tailings by the continuous process, lessens or dispenses with grinding in the pans, decreasing the wear of castings and fuel consumption ; decreases the losses of quicksilver ; increases the capacity of the battery by permitting coarser crushing ; raises the percentage of extraction, and gives higher grade bullion.

The Montana Company by adopting this process obtained, it is said, an increased saving over ordinary pan treatment of from £1 13s. 4d. to £2 1s. 8d., decreased the loss of mercury to $\frac{3}{4}$ lb. per ton of ore, and increased the tenure of the bullion from ·500 to ·900 fine. The cost of the process is from 12s. 6d. to £2 1s. 8d. per ton, and less water is required than with ordinary pan-amalgamation.

It is sometimes more expedient (depending on the base metals present, and the way the ore breaks) to crush finer in the battery and grind less, as before mentioned.

Since ore will break naturally where there is most mineral, by disengaging it as far as possible in crushing, as mercury has a greater affinity for the precious than for the base metals, there is less likelihood of its becoming foul and inactive through taking up lead and other things which grinding tends to make it do.

In Arizona the loss in melting bullion averaging ·938 fine (from volatilization and skimmings) is stated by Mr. Egleston to be 7·55 per cent.

Ores, like some of those of the Comstock, that contain their gold and silver more or less free, will mill from 60 to 80 per cent. by the Washoe process and in exceptional cases, like those of White Pine (Nevada) and Silver Reef (Utah) to even 85 per cent. Roasting milling, on the other hand, gives an extraction of from 80 to 93 per cent., and in exceptional cases 94 per cent.

The combined process yields often 75 to 85 per cent.

The Patio Process.

This very ancient and interesting process cannot be passed by without comment, as Mexico, which is its home, so to speak, is, it must be recollected, the second largest silver producer in the world, and it is safe to say that three-fourths to seven-eighths of the silver it returns is obtained by the patio process.

The practice at San Dimas (State of Durango) may be considered typical, and is excellently described by Mr. R. E. Chism,* whose paper contains detailed information on the subject. Englishmen and Americans alike have gone into Spanish America from time to time expecting to revolutionize this time-honoured system by introducing more modern metallurgical methods, but in most cases they would appear to have failed in doing so, leaving abandoned reduction works all over the country as monuments of their errors. If we look for an explanation, it is to be found in the neglect of local conditions on which the success or failure of every process more or less depends.

The patio, with its cheap plant and fairly close extraction in Mexico, where saving of time is a secondary consideration, where transportation is difficult, where the ores are rich, and where animal power, space, and labour are cheap, whilst fuel and other necessaries are dear (Central America possessing a suitable climate, and labour accustomed to the work from time immemorial) can, in fact, hold its own for the treatment of ores, which do not contain more than a trace of lead and zinc, where other processes would fail.

The silver, which the patio yields, is almost free from impurities and baser metals. An assay of several bars showed an average fineness of 0·994 silver and 0·035 gold. Mr. Chism gives the cost of treatment in a large hacienda where the tahonas were in two groups, and were worked by gear connected with an over-shot waterwheel, where the breaking was done by wooden stamps shod with iron (also driven by a waterwheel), and where the washing was done by a water-power washer, working a trilla of 19 tons, as follows :—

	Cost per Ton of 2,000 lbs. £	s.	d.
Breaking, grinding, and cost of tools	1	7	9
Amalgamator's wages	0	6	11
Scraping tahonas	0	0	8
Carrying and washing scrapings	0	0	5½
Concentrating tailings of scrapings	0	0	3½
Carrying slime from tahona to patio	0	1	9
Mules and keep	0	15	6
Labour, spreading trilla and mule driving	0	6	8
Labour, washing trilla	0	2	4
Charcoal, for retorting silver	0	1	11¼
Concentrating tailings of trilla	0	8	7
Materials— £ s. d.			
Salt, 600 lbs. at 4d. ... ... 10 0 0 ...	0	10	6½
Sulphate of copper, 125 lbs. at 1s. 0½d. 6 10 2½ ...	0	6	10½
Precipitated copper, 25 lbs. at 2s. 9d.... 3 8 9 ...	0	3	7½
Quicksilver, 133 lbs. at 2s. 7¼d. ... 17 6 4 ...	0	18	2½
Total cost	£5	12	1½

* *Trans. Am. Inst. Min. Eng.*, vol. xi., page 61.

The cost to the owners would probably not exceed £5 4s. 2d., as the above costs include a certain charge for profit.

At works where the tortas are of small size (about 10 tons) and the ore is broken without the aid of water-power, inclusive of superintendence and interest on plant, the cost is said to be £5 14s. 11d.

The concentrates obtained from the planilla-treatment and washing in the boliche are shipped and smelted in Germany.

From 70 to 75 per cent., and in some few cases perhaps 80 per cent., of the assay value of the ore in silver is saved by the patio process, 72 per cent. being probably about the average; whilst of the gold present 40 per cent. is lost, 20 per cent. of the remainder goes with the silver, and the rest is recovered from the tailings or is caught in the tahona.

The process from start to finish takes from 23 days to 7 weeks.

In *The Engineering and Mining Journal*, New York, of May 7th, 1892,* Mr. E. du B. Lukis describes, what he claims to be, a recent improvement on the patio system specially applicable to ores containing antimonial and arsenical sulphides of silver, together with ordinary sulphides and some chlorides.

The ore, instead of going direct to the patio after crushing, is first roasted for 10, 12, or 20 minutes. The object of this roasting is merely to give a quick start to the process, and while, owing to its rapidity, it is said not to entail a loss of over $1\frac{1}{2}$ to $2\frac{1}{2}$ per cent. of the silver by volatilization; comparative trials have, it is asserted, proved the benefit to be derived therefrom.

It is not desirable to roast sweet, but only to break up the molecular affinity of the particles of the mineral by heat, so as to hasten the progress of the after-chemical process with a view to save labour and time.

The most important part of the modification in treatment, however, lies in the use of hyposulphite of soda, for if roasted ore be treated by the ordinary patio method it will not work, the quicksilver gets hot at once, and nothing can be done with it.

It is also claimed that it is now possible to arrive at the quantity of sulphate of copper that will be required to beneficiate the cake from the start, reducing the treatment nearly to a certainty. To avoid the usual loss of quicksilver, the amalgamation is stopped before the difficulty of the patio treatment commences (when in fact the free silver ores have been amalgamated), the amalgam is then washed out in the usual way, and the more refractory silver minerals are concentrated. The results are stated to have been satisfactory, showing an extraction of 92 per cent. of the assay value.

* Page 496.

The ores taken for trial were of a mixed class. One-third of the silver being antimonial or ruby silver, the remainder mostly silver glance, but with pyritiferous silver ore and some chlorides. The assay value was 50 dollars per ton. The heavy minerals amounted to about 10 or 15 per cent. of the whole.

The ores were stamped dry to pass a 60 mesh screen, and roasted in an ordinary reverberatory furnace with a hearth measuring 8 feet by 6 feet. The temperature was raised to a good red heat. Charges of 400 lbs. were mixed with $2\frac{1}{2}$ per cent. of salt, spread over the hearth and kept gently stirred with a rabble. Eight minutes after charging, a sample was taken out and washed in the tentadura or assay horn to show the colour of the concentrates. It is by the colour that the calciner knows when to stop roasting. When sufficiently roasted, which does not in any case take more than 20 minutes, the charge is quickly withdrawn, and a fresh one introduced.

The hot ore is allowed to remain in a pile until next day, when it is spread on the patio, moistened with water, and trodden into a soft pulp, with an addition of $\frac{1}{2}$ per cent. of salt. This being done in the morning, sulphate of copper can be added three or four hours later. The required quantity can only be ascertained by experiment, but in the case cited $4\frac{3}{4}$ ounces per 25 lbs. were found the right quantity. After this addition the pulp is again trodden and left till next day.

Early next morning a guide is taken from the cake by a peoñ who walks across it in two diagonal lines taking small portions of the pulp here and there, and accumulating a sample as near as can be guessed of about 100 lbs. This is placed in a corner of the patio, and divided into four portions of 25 lbs. each, to each of which different quantities of sulphate of copper are added, say 1, 2, and 4 grammes (15, 30, and 60 grains) respectively. Assays of each are taken in turn (commencing with the pulp of the cake itself), and mixed up with a globule of mercury in the tentadura and washed, when the action of the sulphate on the mercury will be seen by its colour. It ought to remain bright and quick, and sulphate can be added in small quantities to the guide until the mercury shows a trace of heat, *i.e.*, becomes discoloured and leaden in tone, which indicates that a gramme (15 grains) too much per 25 lbs. has been added.

The amalgamator now knows that he may add 1, 3, or 5 grammes (15, 45, or 75 grains) as the case may be per 25 lbs. of cake. Done with care and patience, this should be a sure guide as to the total quantity of sulphate the cake can stand.

If the process were now continued, as with raw ores, the mercury would become very hot by next day, and soon not work at all, that is to say become dirty in appearance and cease to form silver amalgam. To avoid this trouble, hyposulphite of soda has been used and found to work successfully, whilst it helps to hasten the process. It must, however, be employed with great care, the quantity being found by trial for determined mixtures of ore, and upon this depends its success. Roughly speaking, ½ ounce per 100 lbs. of pulp will be sufficient, more would destroy the mercury.

Having decided upon the additional quantity of sulphate of copper that can be added on this the second day, and having trodden it into the pulp thoroughly, the hyposulphite of soda may be added and trodden in the same way, and immediately afterwards the mercury should be sprinkled over the cake and trodden in, in such proportion as to take up by amalgamation two-thirds of the assay value, or 4 ounces of mercury per ounce of silver present.

The following or second day after incorporating the mercury, treading with horses, spading over, and taking assays will be all that is needed, unless it is found that a guide of 25 lbs. can stand an addition of sulphate of copper. This should not be necessary if the quantity be properly determined before adding the hyposulphite. More than half the mercury will be converted into amalgam by the evening.

The following or third day after incorporating, the same work is required, paying more attention to the assays, and if necessary adding more sulphate of copper, and by evening more than three-quarters of the mercury will be taken up.

The fourth day, the treading, etc., go on, and by evening the mercury should all be converted into silver amalgam, and should be bright and dry, and free from a straw-colour that indicates loss. The next morning the baño or bath is added, consisting of 1½ ounces of mercury per ounce of silver in the cake, which collects the fine amalgam.

This has to be done quickly, and the washing of the pulp follows immediately. The amalgamd, is collecte pressed, retorted, and the bullion in due course melted down.

This treatment extracts that portion of the silver which cannot be concentrated, and it is found easy to concentrate the remainder on vanners, or other machines, obtaining, if necessary, two grades, the first assaying 1,500 ounces or over per ton, and the second 40 to 60 ounces. The waste is then too poor to treat further.

These trials were made with an average temperature of 70 degs. Fahr. in the shade.

The loss of mercury, including the consumido and mechanical loss is said to be about 18 ounces for every 16 ounces of silver, as compared with 22 to 24 ounces lost in the ordinary patio. The gold is collected, if present, partly in the amalgam and partly in the high-grade concentrates. The cost of working a ton of ore assaying 50 dollars is estimated at £1 5s. 10d., extracting 85 per cent. of the silver.

The Fondo and Tina Processes.

In South America, the patio process seems to have been used up to about 1830, when, in consequence of the large quantities of negros or sulphide ores, which began to be found there, it was superseded by the fondo or calderon method, with variations known as the tina, Francke-tina, etc.* The original fondo process was invented, it is believed, in Chili, in the year 1609, by a priest, Albaro Alonzo Barba,† and was used for rich surface-ores, chlorides, bromides, and oxides, which, if not rich enough, required to be concentrated in the planilla, but it was on the whole a very slow process.

The tina process in a modified form‡ is applicable to all ores of silver except argentiferous sulphides of copper, galena, or blende, and to ores which contain more than 1 per cent. of free arsenic, which causes great losses of mercury. The inventor is not known, but it has been in constant use about Copiapo since 1862.

Prof. Egleston§ says of it:—"The whole operation is very simple, quicker with much less loss than the barrel, more certain in its reactions than the patio, and is applicable to almost all the ores found in Chili. It is even cheaper under some circumstances than the lead-fusion."

The cost of treating a ton of £8 6s. 8d. ore, not including interest on sinking fund, working 8 tons a day, is about £1 8s. 7½d.

The process now used in Bolivia, a modification of the tina and fondo processes, seems to be an advance on the former methods, as it includes many of the best points of the tina and pan processes, and may be used for base as well as docile ores of a great variety of yield.

It is closely related to the Francke-tina method, which has been described by Mr. Edgar P. Rathbone‖ and Mr. Arthur F. Wendt.¶

Mr. Wendt says of the Bolivian method, "That large losses of silver were experienced through volatilization when the ore was roasted with

* Egleston, *Metallurgy of Silver*, page 312.

† Percy, *Metallurgy of Gold and Silver*, part I., page 656.

‡ *Revue Universelle des Mines*, series 1. vol. xxxi., page 493.

§ *Metallurgy of Silver*, page 323.

‖ *Proc. Inst. Mech. Eng.*, 1884, page 257.

¶ *Trans. Am. Inst. Min. Eng.*, vol. xix., page 74.

salt in revolving mechanical furnaces, and these could only be obviated by an oxidizing roasting in reverberatories (following lump roasting in kilns), adding the salt after complete oxidation has taken place."

The extraction remains practically the same, whether the chloridization of the ores is 20 or 90 per cent. This is so entirely different from the generally conceived notion of silver-milling in the United States that too much emphasis cannot be laid on this point.

All classes of ores can be worked by the process, not even excepting galena, although the extraction, with the latter and when blende is present, is of course reduced. The bullion obtained is generally ·900 fine or over, alloyed principally with copper. Treating crude ore which carries 75 to 80 ozs. the tailings average about 10 ozs.

The use of bluestone in pan-amalgamation with the object of producing a sub-chloride of copper, which is an active agent in the pan process, has been commented upon before, but when an iron pan is used this sub-chloride is being constantly destroyed. When a copper or bronze vessel, however (like the tina), is used, the sub-chloride is, on the other hand, being constantly regenerated, and this is the essential difference between the two methods, although of course the same results can be obtained at the expense of extra bluestone in the pan in dealing with base ores.

The choice between the tina and the iron pan process must, therefore, Mr. Wendt thinks, depend on the relative price of sulphate of copper and iron-castings as compared with copper, to which must be added the extra refining charges, carriage, etc., on more or less coppery bullion.

The cost at the Real Ingenio Potosi is stated to be 90 Bolivians (£4 13s. 9d.) per cajon of 5,000 lbs., or about £1 17s. 6d. per ton, treating 8 tons per day, but this is, of course, no criterion of what could be done with cheaper freight and better labour in works of large capacity.

The process gives an extraction of 80 to 85 per cent. in Bolivia, and probably on rich chloride ores like those of Caracoles (worked by the Kroehnke process) it would yield returns running up to 92 to 96 per cent., as the extraction on roasted ores averages 90 per cent.

The Leaching or Lixiviation Process.

This process consists in first roasting the ore with salt to convert the silver into chloride, then dissolving the chloride of silver in a solution of hyposulphite of soda and precipitating it with sulphide of lime or soda (as a sulphide of silver) and refining this latter product. The Russel process is a modification of this, consisting in the use of what is known

as extra solution (prepared by adding a certain percentage of sulphate of copper to the ordinary hyposulphite solution) after the ordinary solution has extracted all the silver it will take out.

Lower cost of plant and in some cases less expense in treatment are advantages which are claimed for leaching treatment, but against these must be set the difficulty of obtaining men with practical experience and chemical knowledge necessary to conduct the process, and the fact that the reactions involved are often obscure, and sudden disturbances in working, may be introduced by changes in the character of the ore.

Since lixiviation was introduced into Mexico by Mr. Ottokar Hofmann, in 1868, it has become extensively used, and has risen to a position of importance in some parts of America for the treatment of refractory silver ores. All silver ores are capable of being treated by it with a certain amount of success, except those which contain so much lead that they are classed as smelting ore, or which on account of a clayey-calcareous or talcose gangue, do not permit of free filtration. This latter objection can, however, be overcome, if other circumstances admit of it, by a previous slight concentration. The ore must first be pulverized to fit it for the process, and though this is purely a mechanical operation it does not follow that the machine which will crush the largest amount of ore at the least cost is necessarily the best to select. One has to consider as well in this connexion, the physical condition in which the ore leaves the machine, as on this depends to a great extent a most important part of the process—the roasting. Ores which contain a considerable amount of argentiferous galena or zinc-blende require to be pulverized very fine in order to ensure a satisfactory chlorination, while a coarser pyritic-pulp (whether consisting of iron or copper pyrites) will chloridize well. In selecting machinery for the purpose attention must therefore be paid (as in every other process) to the nature of the ore.

The choice of crushing machinery, for an ore requiring leaching, lies mainly between rolls and stamps. The former produce a much more uniform grain than the latter (the ore and gangue-particles being of more even size), which is of distinct advantage for concentration, but may be the exact opposite for roasting. The most suitable condition of the pulp for roasting in dealing with ore of the kind described, is, in fact, a mixture of fine ore-particles in a coarse gangue, a condition of pulp which is promoted by dry-crushing with stamps. This is explained by the fact that the ore, as a rule, having a higher specific gravity than the gangue cannot so readily escape comminution as the lighter particles of rock, and is only discharged from the screen when reduced to a condi-

tion of comparatively fine pulp, while the lighter gangue is expelled when much coarser.

Roasting.

In roasting, the ore undergoes a chemical decomposition, and this can frequently be better accomplished if the ore-particles are of fine size. The fine condition of the pulp does not seriously interfere with the subsequent lixiviation, because most ore after a chloridizing-roasting (chlorination) becomes sandy, and will filter freely if the gangue is not of a clayey nature. In order to convert the silver into chloride, as before mentioned, the ore must be roasted with salt; and to roast successfully, a sufficient percentage of sulphides must be present, as they produce by their decomposition sulphuric acid (which acts on the salt and liberates the chlorine necessary for the formation of silver chloride). The base metals in the ore are converted into oxides, chlorides, and sulphates. The requisite amount of salt (depending on the character of the ore) varies from 4 to 10 per cent. The salt is either added to the ore before it enters the furnace, or after it has reached a certain stage of oxidizing-roasting. The selection of the proper roasting-furnace to employ is a matter of great importance: this was proved by Mr. Hofmann in Mexico, in experiments made on the San Francisco del Oro ore, in 1888.

It is to be remarked in favour of the Stetefeldt roasting furnace that it requires less salt than any other for chloridizing-roasting silver ores: the decomposition of the salt is very perfect, the chlorine and chloridizing gases emanating from the roasted ore at the bottom of the shaft, acting on the falling ore. Ores which are free from lime and magnesia, can be chloridized in this furnace with a minimum of salt, and it seems to be specially adapted to roasting ores containing copper. Mr. Aaron states that at the Surprise Valley mill, California, an average chlorination of 92 to 93 per cent. was obtained during a nine months' run, roasting silver ores of £15 12s. 6d. assay value, with only $2\frac{1}{2}$ to 3 per cent. of salt. Generally, 5 to 8 per cent. of salt is required in the Stetefeldt furnace, if the ore carries lime and magnesia or a larger percentage of base sulphides. The chlorination varies with the character of the ore, and the attention with which the furnace is managed; results as high as 97 per cent. have been obtained, while they generally range from 87 to 93 per cent. Ores free from sulphur, or with only a slight percentage, should be mixed with 1 or 2 per cent. of iron pyrites, otherwise the salt cannot be decomposed. Oxidized ores carrying peroxides of manganese and iron, which give off oxygen, can be successfully chloridized by themselves. The best results are obtained by mixing oxidized with sulphide ores, more parti-

cularly if the former contain peroxide of manganese. The presence of copper is very favourable for the chlorination of the silver, and if the ore is of such a character that it will bear a high heat without sintering, the chloride of copper formed in the upper part of the shaft can be almost entirely decomposed, and very fine bullion produced by amalgamation. At the Surprise Valley mill, for example, the ore roasted at a low temperature gave bullion only 300 to 400 fine by amalgamation, the base metal being copper. By roasting at a high temperature the bullion was almost freed from copper, its average fineness being 980, running for nine months. All antimonial ores are chloridized with great facility, and with a good system of dust-chambers, the loss of silver by roasting is hardly perceptible. The same is the case with zinc-blende. In roasting ores entirely free from, or with a small percentage of, sulphides, the want of sulphuric acid must, as before remarked, be remedied by adding another substance. A cheap substitute for pyrites is found in green vitriol (sulphate of iron) of which $1\frac{1}{2}$ to 3 per cent. is added when 8 to 10 per cent. of salt is used. The copper is first calcined to drive off its water of crystallization by a gentle heat, and the above percentage of the calcined material is taken. The sulphate then acts on the salt as if it were created in roasting; copperas may also be added to arsenical ores free from sulphurets. If there is a great deal of lime in the ore, it takes up sulphuric acid, forming sulphate of lime, which remains undecomposed; ores containing lime require, therefore, a larger proportion of copperas or iron pyrites, sufficient to transform all the lime into sulphate. At the same time, lime assists in decomposing the base metal chlorides in roasting (5 to 6 per cent. being sometimes added to base ores in a pulverized condition), whilst it does not attack the silver chloride; but too much must not be added if the ore is to be amalgamated afterwards. The lime should be added towards the end of the roasting, introducing 2 per cent. to commence with, and well mixing it with the ore.

Usually, ores containing not more than 8 per cent. of sulphur roast well in the Stetefeldt furnace, but in operating on the San Francisco del Oro ore the results turned out unsatisfactory. This ore contained 25·5 per cent. of zinc, 11·56 per cent. of lead, and 21 per cent. of sulphur, and proved too refractory for the Stetefeldt process. The ore (when sifted into the furnace) created by the sudden combustion of the sulphides an extremely high temperature in the upper portion of the shaft, which caused the suspended ore-particles to slag to globules (a coating of silicates being formed immediately around them), which prevented their further oxidation and chlorination. An analysis showed that 8·48 per cent. of the sulphur, in

fact, remained unoxidized. It was found also that a separation of the ore took place in the furnace, the shaft receiving the portion carrying most of the lead, while the lighter portion, containing most of the iron pyrites, was carried over into the descending-flue, where, owing to this circumstance, combined with the high temperature, that particular portion of it was better chloridized than the bulk which passed through the shaft. An excess of salt lowered the chlorination, and 12 per cent. was found to give the best results (60·8 per cent. chlorination) in the descending-flue. Another bad feature was the formation of lumps and crusts in the upper region of the shaft, and choked the lower side of the feeding-screen.

An ore of the class operated upon requires to be submitted to a long and gradually increasing temperature before the salt is added ; and as the principle of the Stetefeldt furnace is just the reverse of this, the results could not but be unsatisfactory. Heavily sulphide ores, especially if they carry zinc, require more draught in roasting than lighter sulphide ores ; and this is particularly the case when they are exposed for such a short time, as in the Stetefeldt furnace, to the action of air and heat. Mr. Hofmann's experiments in roasting the ore of the San Francisco del Oro mine in a Stetefeldt furnace go to show :—

1. An incomplete oxidation of the sulphide minerals, the main portion of the ore still containing 8·48 per cent. unoxidized sulphur when roasted with salt, and 7·6 per cent. when roasted without salt.
2. An insufficient chlorination of the silver. The highest chlorination of the ore in the shaft was only 16·9 per cent., and as 62·5 per cent. of the whole volume dropped into the shaft, the somewhat higher results obtained in the descending-flue could not much improve the average.
3. That the principle of the Stetefeldt furnace is contrary to the conditions, the maintenance of which are so essential to roasting ores containing much zinc blende and galena.
4. That, on acount of the sudden exposure of the raw ore-particles to such a high temperature, they melt to minute globules, which make the ore unfit for further treatment.
5. That a concentration of the lead minerals takes place in the shaft, which is disadvantageous.
6. That about 25 per cent. of the ore, when passing through the furnace, is changed into hard lumps of almost raw ore.
7. That the lower side of the feed-screen becomes rapidly encrusted and the holes obstructed, requiring frequent changes of screens.

The ore after passing through the shaft of the furnace had a dark, almost black colour, and continued to emit volumes of sulphurous acid gas when discharged, and for some days after, but no chlorine. Mr. Hofmann next tried re-roasting the partly roasted ore from the Stetefeldt in a modified Howell furnace. Having previously ascertained that this ore contained only 1·3 per cent. of salt, 6 to 8 per cent. more was added, and the feed regulated so as to put through from 5 to 9 tons per 24 hours. The average of thirty-three charges worked thus gave :—average of re-roasted ore, 31·42 ounces of silver per ton ; average of leach-tailings, 17·55 ounces per ton ; average of chlorination, 44·2 per cent.

It is a point to note that the consumption of wood in re-roasting proved to be much greater than in roasting raw ore. This is accounted for by the main portion of the sulphur, combined with the pyrites, having been burnt off in the Stetefeldt furnace. During the second roasting, the furnace was deprived of the heat of combustion derived from this source, so that additional extraneous artificial heat had to be furnished. To re-roast in this manner 8·3 tons, it took 26 cargas of wood (12 cargas = 1 cord), while it took only 16 cargas to roast 10 to 11 tons of raw ore. The roasting capacity of the furnace was also diminished. By dumping the red-hot ore from the Stetefeldt direct into a reverberatory hearth, the extra consumption of wood during the finishing roasting could doubtless have been materially lessened ; but what made this plan impracticable was the change the ore underwent in passing through the Stetefeldt furnace, which interfered with attempts to obtain a high chlorination result afterwards.

The re-roasted ore was of a red-brown colour, smelled of chlorine and did not emit any sulphurous acid gas, but it still consisted principally of the little globules, of which mention has been made, quite a large number of which remained black, no matter how long the ore was kept in the furnace. Some were magnetic, but the majority were not. Between the fingers, the re-roasted pulp felt sharp, like powdered glass. The temperature was kept at a proper degree, and there was an abundant draught, the dust-chambers and furnace having been previously cleared. Still it was impossible to obtain more than 44·2 per cent. of chlorination, owing undoubtedly to the silicates formed during the roasting in the Stetefeldt furnace. A jet of steam* introduced into the reverberatory hearth attached to the modified Howell furnace considerably improved the results, but still did not give sufficient satisfaction.

* The introduction of steam through the bridge of the furnace generally increases the fuel consumption.

The average of thirteen charges showed:—average of re-roasted ore, 31·0 ounces per ton; average of leach-tailings, 10·37 ounces per ton; average of chlorination, 66·6 per cent. The ore still contained a considerable number of globules, but they had changed their colour to red-brown, and between the fingers the ore felt soft and not so sharp and glassy as when roasted without steam. The Howell furnace alluded to above was not a regular Howell, but consisted of a revolving cylinder of uniform diameter with a shell of boiler-plate lined with bricks throughout its length (the principle on which it works is identical, however, with the Howell furnace), and had a reverberatory hearth in front, attached at the end.

The Howell, like the Stetefeldt furnace, requires the salt to be mixed with the ore before entering the furnace. With some ores this is immaterial, but it is a matter of the greatest importance for the del Oro ore. If the salt is previously added the ore becomes sticky, encrusts the furnace rapidly, and leaves the furnace mostly in lumps imperfectly chloridized. If the ore is charged without salt it remains dry and sandy, but a very annoying separation takes place. The fine particles are carried by the draught into the dust chambers, and only the coarse sand passes through the furnace without being sufficiently desulphidized. If, then, salt be added in the drop-pit, only a small percentage of silver becomes chloridized. The best results gave only 29 per cent. of chlorination. In order to diminish the separation, 2 per cent. of salt was added to the ore in the battery. This small percentage of salt made the ore sticky enough to diminish considerably the dusting, without causing the formation of lumps or too heavy encrustations.

By this method, the chlorination improved considerably, the average of three days' run being 67 per cent., but Mr. Hofmann convinced himself notwithstanding, that the Howell furnace, as such, could not roast the del Oro ore. The results were not sufficiently uniform and reliable; and though the roasted ore was left in a good condition, the average could not be brought above 67 per cent. An alteration being apparently required, which would give the ore more roasting time and allow of a better regulation of the temperature, Mr. Hofmann made the following changes. In front of the furnace, he constructed a shallow drop-pit and a fireplace, and on one side of the drop-pit, communicating with it, he built a small reverberatory hearth, 6 feet by 8 feet, the bottom of both being on the same level.

The reverberatory hearth contained one working-door and a 24 inches fireplace. When enough ore had accumulated in the pit to make a charge for the reverberatory hearth, it was pushed in with a hoe, each charge consisting of about 1,400 lbs. When starting the furnace, a strong fire

was kept up in both fireplaces, but after the process was in operation the fire in front of the cylinder was much lowered ; in fact so much so, that half the grate bars remained bare of wood. Only now and then a thin stick of wood was added, just enough to prevent the drop-pit getting chilled. Two per cent. of salt was added to the ore in the battery.

If the roasting be properly conducted, the blue flame of the ignited pyrites can be observed in the back part of the cylinder. Next to it, and reaching beyond the middle of the cylinder, the ore assumes a higher temperature, forming a belt of bright red heat. The part of the furnace next the fire (nearly one-third of the whole length) should look dark. The furnace is mostly heated by the combustion of the sulphides, and receives but little supply from the fireplace and reverberatory hearth ; in fact the ore in the cylinder should be left as much as possible to roast in its own heat. This is a very important condition to maintain, the object being to convert as much as possible of the galena and zinc-blende into sulphates and oxides before generating chlorine, and to avoid until then as much as possible the decomposition of the iron-salts. This can only be done by maintaining a low heat after the combustion of the pyrites.

An excess of heat is invariably connected with an excessive loss of silver by volatilization and by a low chlorination. Galena and zinc-blende roast quicker and better in a low than a high temperature. When the ore leaves the cylinder and drops into the pit, it should be of a dull red heat, while the colour after cooling should be dark yellow-brown. If the temperature be so kept, neither the odour of chlorine nor much of sulphurous acid can be detected. At an increased heat, sulphurous acid is again evolved strongly, showing that the oxidation is not yet completed. As the temperature in the cylinder is mostly produced by the combustion of the sulphides, the chief means of regulating the same is the feed. If too much ore enters the furnace, the belt of bright red heat increases, advancing more and more towards the front, and finally the whole furnace assumes this temperature. The ore dropping into the pit is very hot, emits heavy fumes and overheats the pit. If it be then removed into the reverberatory hearth, it takes a very long time to finish, necessitating an interruption in the feed of the cylinder.

On the other hand, if insufficient ore be charged, the belt of bright red heat gets smaller and moves towards the back end of the furnace. When the properly-prepared ore enters the reverberatory hearth, the salt is added and the temperature increased. It commences to fume, and swells without forming more lumps than an ordinary ore. In the beginning, strong fumes of sulphurous acid are emitted, but soon cease and chlorine appears.

The charge is finished if the fumes assume a mild and sweetish smell of chlorine; as long as they smell strong, roasting must be continued.

Mr. Hofmann made a series of experiments to ascertain the smallest amount of salt practicable, and found that 4, 6, 8, and 10 per cent. gave about equal results; 12 per cent. commenced to make the ore too sticky and produces less chlorination; 3 per cent. was sufficient if the roasting were very carefully conducted, but then only 1 per cent. had to be added in the battery and 2 per cent. in the furnace; 4 per cent., however, was safer, for then the result did not depend so much on the skill and goodwill of the labourers. The roasting capacity of the furnace proved much less for the del Oro than for ordinary ore. The cylinder was only 24 feet long, which could not roast more than $8\frac{1}{2}$ tons per 24 hours; but even with a 32 feet cylinder, Mr. Hofmann states he did not expect to roast more than 12 tons per day.

Each charge had to remain 2 hours on the reverberatory hearth. Though the ore was roasted in the reverberatory hearth at a somewhat increased heat, it could not be raised beyond dull red without losing too much silver by volatilization.

If the silver-bearing minerals of an ore be not of great density and decrepitate in the heat, the ore can be crushed coarse without endangering the subsequent roasting, but if the principal silver-bearing mineral, like the zinc-blende in the del Oro ore, be of great density and does not decrepitate, it is of the greatest importance to crush fine.

A series of experiments was made by Mr. Hofmann with ore crushed through No. 20 and No. 40 screens. He found that the ore crushed through No. 20, required a much longer time and was 27 per cent. less chloridized than the ore crushed through No. 40 screen. The material which passes through a battery-screen of a certain number, is much finer than the size of the meshes. Heavy ore makes a much finer pulp through the same screen than lighter ore. The pulp of the del Oro ore obtained by crushing through battery-screens Nos. 20 and 40, when sieved through sieves of different fineness, showed the following results:—

Battery Pulp when Sifted through Sieve. No.	Crushed through Battery Screen No. 20. Percentage of Material passing through the Sieve.	Crushed through Battery Screen No. 40. Percentage of Material passing through the Sieve.	Crushed through Battery Screen No. 20. Percentage of Material remaining on the Sieve.	Crushed through Battery Screen No. 40. Percentage of Materia remaining on the Sieve.
30	93·8	100	6·2	0·0
40	87·3	100	12·7	0·0
60	78·8	98·95	21·2	1·05
80	71·2	93·80	28·7	6·20
90	67·1	90·50	32·9	9·50

These figures show how exceedingly fine a heavy ore is crushed (dry) in a battery, even through a screen with comparatively coarse meshes. Though 67·1 per cent. of the material crushed through screen No. 20 was finer than sieve No. 90, the average chlorination of quite a number of comparative roastings was 27 per cent. less than that of ore crushed through a No. 40 screen. This indicates how essential it is to crush such ores fine.

Mr. Hofmann found that in order to chloridize the del Oro ore well, it had to be crushed through a No. 40 battery-screen, which furnished a pulp of which only 9½ per cent. was coarser than sieve No. 90. To produce such a fine pulp with rolls, would require such exceedingly fine screens that their use would be here impracticable. The crushing capacity of the battery was not much diminished by using No. 40 screen instead of No. 20, principally because the ore was so heavy.

For comparison, Mr. Hofmann ran 5 stamps, 12 hours, crushing through No. 20, and 5 stamps through No. 40, and found :—

No.	Lbs. of Pulp.
20 with salt furnished...	8,100
40 with 4 per cent. salt furnished ...	7,488
Difference in favour of No. 20 screen	612 lbs., or 7½ per cent.

A very small reduction, if the great advantage gained for roasting is considered. Some ores gain much in chlorination of the silver, if left hot in a pile for several hours; this is mostly the case when an ore is insufficiently roasted, or when the nature of an ore is such as to require long roasting at low heat.

Additional chlorination can be produced by moistening the ore and leaving it for several hours in the pile; this is usually the case if the ore contains copper. Roasted ore containing caustic lime should not be left moist on the cooling-floor, hence the del Oro ore could not be wetted on the cooling-floor without decomposing some of the silver chloride. Mr. Hofmann remarks that the most important additional chlorination takes place, however, during base-metal leaching.

In the Silver King ore, Arizona, this additional chlorination amounted to nearly 6 per cent. Mr. Hofmann therefore filled the vat before charging about one-third with water and dumped the ore hot into it, thus producing a hot base-metal solution, and made the observation that by it not only the decomposition of the silver chloride was avoided, but that a considerable increase in the silver chlorination took place, amounting in some instances to 12·9 per cent. If the original chlorination was 75 per cent. or more the increase was, however, much less. By adding some cupric chloride to the water in the vat before dumping in the ore, badly roasted

charges gained as much as 34 to 38 per cent. (see charges, Nos. 9, 15, and 16 in annexed table). These are very important observations, and give the operator a means of correcting badly roasted charges.

The following table is a careful record of the results obtained in roasting the San Francisco del Oro ore in the Howell furnace modified by Mr. Hofmann, representing a two weeks' run. As each tank-charge contained the whole ore of 24 hours' roasting it offered a good opportunity of following each charge through the process from the condition of raw ore to tailings, and of ascertaining for each the loss by volatilization, etc.

Taking the average of the results we find the silver chlorination when the ore left the furnace was 68·4 per cent.; the additional chlorination was 13·3 per cent.; or a total of 81·7 per cent. The low average chlorination of the ore when leaving the furnace, was caused by the three badly-roasted charges 9, 15, and 16. The other eleven charges gave an average of about 75 per cent.

Mr. Hofmann remarks—"The total or actual chlorination of 81·7 per cent. may seem to be low, but if we consider that the ore is of low grade, averaging only 28·85 ounces per ton, and that 1 per cent. represents only 0·28 ounce of silver, and also consider that the ore contains about 37 per cent. of zinc-blende, and 13 to 19½ per cent. of galena, which carry all the silver, and that the ore was pronounced as being too refractory for chloridizing-roasting, we have to count the work done by the modified Howell furnace, as very satisfactory."

In order to ascertain the loss of silver by volatilization we have first to ascertain the loss in weight which the ore sustains during roasting. This can be done by the muffle test described by Mr. Hofmann in *The Engineering and Mining Journal*, New York, of April 23rd, 1887.* "Ten grammes of the raw pulp containing salt are placed in a roasting dish and roasted in the muffle for half an hour or an hour, then the sample is removed from the muffle, allowed to cool, weighed, placed back in the muffle, and roasted again for half an hour, then weighed again. This is repeated till two weighings are alike, or till in the last half hour of roasting, the ore does not lose more than 2 or 3 milligrames. The difference between the original weight and that of the last weight expressed in percentage gives the highest possible loss which the raw ore can sustain."

Ten grammes of a sample of roasted ore corresponding with the sample of raw pulp are placed in a roasting dish, and also roasted in the muffle until two weighings agree or differ after half an hour's additional roasting not more than 2 or 3 milligrammes. The difference between the first weighing and the last (expressed in percentage) gives the weight which the

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WORKING RESULTS OBTAINED IN ROASTING.—EXPERIMENTS WITH THE MODIFIED HOWELL FURNACE.

Date, 1888.	Per Cent. of Salt used.	Value of Raw Ore.	Value of Raw Ore containing Salt.	No. of Tank-charge.	Value of Roasted Ore.	Value after Leaching in the Laboratory.	Per Cent. of Chlorination, when the Ore left the Furnace.	Value of the Vat-tailings.	Per Cent. of Actual Chlorination, Calculated from the Vat-tailings.	Loss in Weight which the Ore Sustained during Roasting.	Loss of Silver by Volatilization.	Loss of Silver by Tailings.	Per Cent. of Chlorination Gained in the Vat.	Number of Ozs. of Silver Extracted.	Per Cent. of Actual Silver Extraction.	Remarks.
		Ozs. per Ton.	Ozs. per Ton.		Ozs. per Ton.	Ozs. per Ton.	Per Cent.	Ozs. per Ton.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Ton.	Per Cent.	
April 20	5½	34·21	32·33	9	31·34	11·66	37·2	7·75	75·3	2½	5·2	24·5	38·1	24·04	70·3*	Assorted ore.
„ 21	6	34·50	32·43	10	30·91	9·04	71·8	7·26	76·6	2½	7·1	21·8	4·8	24·52	71·1	
„ 23	5	35·09	33·34	11	31·49	8·89	71·8	4·33	84·7	2½	7·8	14·1	12·9	27·40	78·1	
„ 24	5	33·76	32·07	12	29·74	8·16	72·6	3·29	89·0	2½	9·5	10·0	16·4	27·17	80·5	
„ 25	5	31·67	30·09	13	29·59	6·99	76·4	3·57	88·0	2½	4·1	11·5	11·6	26·72	84·4	Ore slightly assorted.
„ 26	5	29·09	28·64	14	29·16	7·58	74·0	4·25	85·5	3½	1·7	14·3	11·5	24·55	84·4	
„ 27	4½	27·35	26·02	15	25·95	13·27	51·9	4·25	84·7	3½	3·7	15·7	32·8	22·04	80·6*	
„ 28	4	28·32	27·19	16	24·49	12·89	47·4	4·48	81·8	3½	13·0	15·8	34·4	20·16	71·2*	
„ 29	4	26·02	24·98	17	22·16	4·95	77·7	4·76	78·7	3½	14·2	19·0	1·0	17·38	66·8	
„ 30	5	25·43	24·42	18	22·74	5·24	77·0	4·48	80·3	3½	10·1	18·3	3·3	18·20	71·6	Unassorted ore, just as it was broken-down in the mine.
May 1	5	24·37	23·15	19	21·57	4·66	78·4	4·84	77·6	3½	10·0	20·0	0·8	17·05	70·0	
„ 2	5	23·94	21·97	20	22·45	5·39	76·0	3·87	82·8	3½	1·3	17·0	6·8	19·46	81·3	
„ 3	4	24·28	23·31	21	22·91	5·83	73·6	4·84	78·1	3½	8·7	20·0	4·5	17·31	71·3	
„ 4	4	26·02	24·98	22	21·87	6·12	72·1	4·30	80·4	—	15·5	16·6	8·3	17·66	67·9	
	4·7	28·85	27·44		26·10	7·90	68·4	4·76	81·6	3	7·9	17·0	13·3	21·69	74·9	Averages.

Charges marked * were treated during base-metal leaching with cupric chloride.

roasted ore is still capable of losing if subjected to prolonged roasting. If we deduct this last loss from the highest possible loss, we obtain in percentage the loss in weight by volatilization which the ore has suffered during roasting in the furnace.

Owing to a great part of the lead and zinc sulphides being converted into sulphate, the San Francisco del Oro ores lose but a small percentage of their weight during roasting. The tests showed a loss of $2\frac{1}{2}$ to $3\frac{1}{2}$ per cent. With these figures and the assay value of the raw and roasted ore, the loss of silver by volatilization was calculated. The extremes were 1·7 and 15·5 per cent., and the table illustrates how variable this loss is. Mr. Hofmann says he found the del Oro ore more sensitively disposed for such losses than many others, even antimonial ores. The least increase of temperature above dull red causes a heavy loss, even if this increase of temperature lasts only for a very short time. Two or three thin sticks of wood thrown on the fire before they are needed may materially increase the loss. The loss by volatilization is not in direct proportion to the percentage of chloridized silver. In proof of this Mr. Hofmann instances the results of roasting Las Yedras ore at high and low temperatures in Brückner furnaces :—

High temperature, averages of 31 consecutive working-days, January 13th to February 12th, 1887.

	Furnace No. 3. Per Cent.	Furnace No. 4. Per Cent.	Average of both Furnaces. Per Cent.
Chlorination	71·3	74·2	72·7
Loss by volatilization ...	18·3	17·6	17·9

Low temperature, averages of 30 consecutive days, June 1st to July 1st, 1887.

	Furnace No. 3. Per Cent.	Furnace No. 4. Per Cent.	Average of both Furnaces. Per Cent.
Chlorination	81·7	81·3	81·5
Loss by volatilization ...	1·7	0·7	1·2

	Per Cent.
By comparing these average results we find an increase of chlorination, in favour of low heat, of	8·8
A decrease of loss by volatilization caused by low heat ...	16·7
Total increase of production	25·5

The roasted ore contained only a small percentage of lumps, not hard, but porous, which fall to powder if kept in contact with water for some time. If the ore be left dry in a pile, it hardens. If left undisturbed for a week or two, it becomes so hard that a pick is necessary to loosen it. In water, however, it softens easily again. The colour is usually red-brown,

but occasionally, if the ore contain less iron pyrites, it is yellow-brown. The heavy metals in the Yedras ore are principally converted into sulphates, and only a small proportion are present as chlorides.

The following is an analysis of the unassorted ore after roasting with 5 per cent. of salt :—Gold, traces ; silver, 0·09 per cent. ; lead, 9·00 per cent. ; iron, 6·00 per cent. ; zinc, 22·45 per cent. ; caustic lime, 5·65 per cent. ; antimony, 0·75 per cent. ; copper, 0·60 per cent. ; cadmium, 0·10 per cent. ; alumina, 3·09 per cent. ; caustic soda, 3·79 per cent. ; sulphuric acid, 13·16 per cent. ; chlorine, 0·88 per cent. ; soluble silica, 8·00 per cent. ; and insoluble gangue, 18·61 per cent.

Mr. Hofmann mentions an easy method of removing crusts from continuous discharging-furnaces, which can be done while the furnace is in operation. In the masonry at the back an opening is made, so located that with a long iron spade the interior of the cylinder can be reached just below the feed-pipe. Through this opening, 1½ dozen firebricks are introduced while the cylinder is revolving, and in moving slowly forward shove off the crust clean down to the lining. The crust when hot is soft and yields to the weight of the moving bricks. It is not necessary to interrupt the feeding. If bricks should be found too light, heavier, but not too large, pieces of castings may answer.

Owing to the large quantity of sulphides in the ore and the very low temperature at which the del Oro has to be roasted, the consumption of wood is very small. After the furnace is heated and the cylinder encrusted, it takes hardly any fire in front of the cylinder to maintain the proper temperature. In the reverberatory a little more fire is needed, but much less than with ordinary ores. Mr. Hofmann states that he weighed the wood during a couple of weeks, and found the total consumption during this period to be 220 cargas (at 300 lbs.). With this quantity of wood he roasted 115·8 tons, and therefore used 1·8 cargas per ton. 12 cargas of Parral wood being equal to 1 cord; if we express the consumption in cords, we find that with 1 cord of wood the furnace roasted 6·3 tons of ore. The cost of roasting in the modified Howell furnace (roasting 8½ tons per 24 hours) is as follows :—

	Dollars.
2 head roasters at $1·50	3·00
4 helpers at 90 cents	3·60
4 per cent. salt, 680 lbs. at 1·27 cents ...	8·63
15·7 cargas wood at 75 cents	11·77
Steam-power, 10 cargas of wood at 75 cents	7·50
Oil, light, tools, etc.	2·00
Management, office, mechanic, assay office	1·78
Total	38·28

And the cost per ton, $4·50, Mexican currency.

To ascertain the cost of steam-power a separate boiler was used for the furnace. It is evident that by using a boiler for only one small furnace, the expense per ton of ore will be much greater than if with the same boiler and engine several large furnaces are operated, but this was the only way of getting an estimate.

The steam for working the pump and preparing the calcium sulphide were supplied by the same boiler and had to be charged to roasting. It is included in the above statement, and will therefore not appear in the statement of cost of lixiviation.

As the statement of cost is made for only $8\frac{1}{2}$ tons per day, it would be misleading if the whole expense for management, mechanics, assay office, etc., were charged to the $8\frac{1}{2}$ tons, as these expenses would remain the same for 100 tons a day—the intended capacity of the new mill. The proportion chargeable on $8\frac{1}{2}$ tons, treating 100 tons per day, has therefore been debited to this item, but as there are three departments in the mill, viz., stamping, roasting, and leaching, each department has to be charged with one-third of this expense; 1·78 dollars represent therefore one-third of the cost.

The figures contained in the preceding table are the results of experiments obtained with different treatment as regards salt, temperature, time, etc., and the averages do not therefore represent the best obtainable results. This is especially the case in regard to loss by volatilization, which should be reduced as the men gain experience in handling the furnaces. They were good enough, however, to secure a profitable reduction of the ore, and may be accepted as a basis for calculations and estimates.

In order to increase the roasting capacity to the stamping power of the mill and for the sake of further experiments, Mr. Hofmann erected four double-hearth reverberatory furnaces (the lower hearth of 220 square feet surface, and the upper of 210 square feet surface) to supplement the modified Howell furnace.

Each double furnace took four charges of 1 ton each; when one charge was finished, all the others were moved forward, and on the first hearth a new charge dropped through the opening in the arch. From the second hearth the charge was dropped on the lower hearth, through an opening in the bottom near the working-door. The upper hearth was exclusively used for oxidizing-roasting, and 4 per cent. of salt was added while the charge was dropped on to the lower hearth. Every $2\frac{1}{2}$ to 3 hours a charge was done, and therefore each double furnace roasted from 8 to 10 tons per day, according to the quantity of lead the ore contained. The chlorination results were so near those obtained in the modified Howell

furnace that it is not necessary to give details, but Mr. Hofmann records the observations he made on a charge submitted to a prolonged oxidizing-roasting. They are too long to be given here, but will be found fully described in *The Engineering and Mining Journal*, New York, of February 23rd, 1889.*

He found that the ore during oxidizing-roasting sustained a loss of silver by volatilization of not less than 13 per cent. (not taking into calculation the loss in weight), and that this loss principally occurred during the seventh hour, at the time the ore assumed a dark red-brown colour.

Only a part of the soluble silver, he concludes, was present as sulphate, viz., 25·2 per cent. Of the 58·8 per cent. which was soluble, 33·6 per cent. was some other silver salt, not soluble in water, but soluble in sodium hyposulphite, probably silver antimoniate.

The reverberatory furnaces were built in pairs, two being connected with one main flue.

During four weeks the wood consumed by one pair of these furnaces was weighed. In this time 507 tons of ore were roasted at a consumption of 672 cargas of wood, or 1·3 cargas per ton. If expressed in cords, we find that one cord of wood roasted 9 tons of ore, an extremely small consumption.

The cost of roasting in the reverberatory furnace, for two double-hearth furnaces roasting 18 tons per day, was as follows :—

	Dollars.
2 head roasters at $1·50	3·00
16 roasters at $1·00	16·00
2 wood carriers and ore wheelers at 90 cents	1·80
2 carmen for raw ore chargers at 90 cents	1·80
23·4 cargas of wood at 75 cents	17·55
4 per cent. salt, 1,440 lbs. at $1·27 ...	18·28
Tools, etc.	4·00
Management, office, mechanics, assays ...	3·77
Total	66·20

And the cost per ton, 67 cents, Mexican currency.

To form an opinion as to which roasting-furnace is the most suitable for an ore like the San Francisco del Oro, we have to take into consideration only the modified Howell and the reverberatory furnaces. The Stetefeldt furnace did not answer, and the Brückner furnace was not tried, as experience has shown that a large Brückner furnace could not roast more than 5 or 6 tons of such a heavy ore. Both the modified

* Page 190.

Howell and the reverberatory furnace gave about the same results, and the loss by volatilization was also nearly the same. The cost, however, was different, viz., 4·50 dollars per ton in the former, and 3·67 dollars in the latter, being a difference in favour of the reverberatory furnace of 83 cents per ton.

This ought to decide; still there are, besides the merits of the furnace, other circumstances to be taken into consideration. The reverberatory furnace requires quite a number of skilled hands, and the result depends much more on the skill and goodwill of the men, than with the modified Howell. Mexicans are, as a rule, if properly handled, very good workmen, notwithstanding that they receive such moderate wages, but they like to lay off on Sundays and feast days. On such days quite a number of inexperienced substitutes will be found working at the furnaces, and the roastings consequently suffer. The main trouble, however, occurs every year in spring and autumn, the time of planting and harvesting corn, when many leave the camp, and the mill is left short of hands. For this reason it may be more advisable in a large mill to use the modified Howell furnace, notwithstanding that the reverberatory furnace works cheaper and creates much less flue-dust. The Howell furnace forms a great deal of flue-dust, which is far from being roasted even if the furnace be provided with an auxiliary fireplace, as in the Howell-White furnace.

Lixiviation.

Base-metal Leaching.—The roasted ore is moistened with water and charged into wooden vats, in charges of 10 to 15 tons, though much larger charges are sometimes worked. The vats are best furnished with a central discharge, around which a filter-bottom is arranged in the shape of a very flat funnel. The filter-cloth is kept in place by ropes driven into grooves around the discharge-hole and inner periphery of the tub, near the filter-bottom. The vat is provided with an outlet under the filter, and has a slight inclination towards this point.

The charge of roasted ore is leached with water to remove the soluble base-metal salts. Water does not dissolve silver chloride, but a concentrated solution of base-metal chlorides does, and it is therefore advisable not to make the leaching-vats too deep, as otherwise a too concentrated base-metal solution is produced by the water in descending through a thick layer of ore. The base-metal leaching is completed when a few drops of calcium polysulphide poured into some of the outflowing solution does not produce a precipitate. This part of the process, according to the character of the ore, takes 4, 8, or 10 hours.

With the San Francisco del Oro ore, which filters well, the rate of filtration during base-metal leaching was 1 inch in 6 minutes or 10 inches per hour. With a vat 10 feet 2 inches in diameter, representing a filtering surface of 81·01 square feet, and a thickness of charge of 2·41 feet, the above rate was equivalent to 67·24 cubic feet, or 504·3 gallons per hour; leaching time, 8 hours = 4,034·4 gallons; 2 feet of water in the vat into which the ore is dumped = 1,215·2 gallons extra. Second washing, rate of filtration, $8\frac{2}{3}$ inches per hour; time, $1\frac{1}{2}$ hours = 647·1 gallons. Total for one charge of 8·39 tons, 5,896·7 gallons, or 703 gallons for each ton of ore. It has been mentioned that, on account of the roasted ore containing considerable caustic lime, it was not moistened as usual on the cooling-floor but dumped dry and hot into about 2 feet of water in the vat. Besides other advantages, this way of charging shortens the base-metal leaching time by nearly an hour, without reducing the rate of filtration. When the charge is in, the water ought to cover the ore about 1 inch. Leaching is then commenced from below,* and the solution diluted above. As soon as the solution is allowed to flow out below, a stream of clear water has to be turned on above and continued for about an hour, in order to produce a greater dilution. Then the influx is interrupted until the water sinks to the level of the ore, when water is again allowed to flow in. The base-metal solution of the del Oro ore shows an acid reaction. The leaching with water is continued until a few drops of calcium sulphide added to the solution produces only faint white clouds. These white clouds continue to show for hours, and are caused by the reaction of calcium sulphide and sodium sulphate precipitating gypsum. It takes a long time to leach alkaline salts contained in porous substances, thus it happens that while all the soluble metal salts have been dissolved and removed, the outflowing solution still contains and continues to contain sodium sulphate.

From a chemical standpoint, it would be advisable to continue leaching with water till all the sodium sulphate is removed, but in practice this would delay the process too much, and the silver leaching is therefore commenced as soon as all the heavy metal salts are removed. The sodium sulphate which still remains in the ore after leaching with water, enters the stock solution during the subsequent silver leaching. This has not a very injurious effect if calcium sulphide is used as a precipitant, because sodium sulphate is decomposed and gypsum precipitated.

The stock solution is therefore freed from sodium sulphate after every

* This is done to precipitate on to, and through the ore any chloride of silver that might be dissolved by a too concentrated solution of the base-metal chlorides.

precipitation, and the only resulting disadvantage will be that the precipitate will contain more gypsum. But if sodium sulphide is used as a precipitant, the effect of the sodium sulphate is very injurious, as it remains and accumulates in the stock solution and soon reduces its dissolving energy for silver chloride, rendering a prolonged leaching with water necessary. By inserting, in the outlet of the vat, a small rubber tube (provided at the end with a glass tube drawn to a fine point), and by leaving this tube in the outlet during the whole time of base-metal leaching, Mr. Hofmann obtained from the outflowing solution a very fine stream which, collected in a proper vessel, gave a true sample of the solution of about 3 or 4 gallons. This sample contained in 1,000 cubic centimetres :—

	Grammes.		Grammes.
Silver	0·0036	Lead	—
Cadmium ...	0·806	Sulphuric acid ...	18·708
Zinc	2·105	Chlorine	7·173
Iron	0·008	Lime	0·754
Copper... ...	Trace		

Silver Leaching.—The base-metal salts being removed, a stream of diluted solution of sodium hyposulphite is allowed to enter on top of the ore, which readily dissolves the silver chloride. When the outflowing solution shows indications of silver, the stream is conveyed to special precipitating-tanks, in which the silver is precipitated as silver sulphide by an addition of calcium polysulphide or some other reagent.

To ascertain the exact time when the hyposulphite solution, which follows the water, commences to appear at the outlet, a matter of importance, it is ordinarily tested with calcium polysulphide, and some operators indulge in the bad habit of testing it with the tongue, and are guided by the sweetish taste which the solution has if it contains silver. The following test is much superior, and is very sensitive and convenient :—A small strip of starch paper is dipped into a solution of iodine and then held in the outflowing stream. If the blue colour disappears it is a sign that the liquid contains a hyposulphite salt, and the stream must be turned into the precipitation-vats. The test is only applicable, however, when the base-metals are leached with cold water, as hot water discolours the blue paper. To facilitate and hasten the settling of the silver precipitates, the precipitation-tanks are provided with machine stirrers, by which the solution can be vigorously agitated.

After precipitation, the sodium hyposulphite solution is decanted after the precipitate has settled in tanks placed at a lower level. From these tanks the clear solution is pumped up to storage-tanks and

is ready to be used again. When all the soluble silver is extracted, the solution of the hyposulphite is allowed to run out till it disappears under the surface of the ore, when clear water is introduced again to displace all the hyposulphite solution; after this second leaching with water, the tailings are sluiced out through the central discharge, and the tank is ready for another charge of ore. The time for silver leaching varies according to the character of the ore from 8 hours to 2 or 3 days.

In the ordinary lixiviation process, calcium sulphide is the ordinary and best precipitant. This salt, however, cannot be used for that purpose in the Russel process, and sodium sulphide therefore takes its place. In the Kiss process the ore is leached with calcium hyposulphite, and the silver is precipitated with calcium sulphide. Leaching with sodium hyposulphite and precipitating with calcium sulphide (the ordinary method) is really a combination of the Patera and Kiss processes. When enough silver precipitate has accumulated on the bottom of the precipitating-tanks, it is drawn off and generally strained through a filter-press. The black silver cakes are then taken out, dried in a warm room or drying-oven, and introduced into a muffle or calcining-furnace to burn off the sulphur. After the blue flame has disappeared, heating must continue for several hours at a dark red heat.

The roasted cakes may be melted with lead in a cupelling-furnace, and refined or melted in graphite crucibles, in lots up to 300 lbs. in weight. What sulphur remains must be removed by melting it off with metallic iron introduced into the pot, an iron matte being formed, which rises to the surface and is skimmed off. The surface of the silver is then cleaned by adding some bone ash and borax, or borax alone, which is also skimmed off, and the silver dipped out or poured into moulds.

The quantity of hyposulphite solution required for treating the San Francisco del Oro ore is as follows :—The diameter of the vat is the same as for base-metal leaching (previously described), viz., 10 feet 2 inches. Rate of filtration, $8\frac{2}{3}$ inches per hour or 57·52 cubic feet or 431·4 gallons per hour. Time of silver leaching, 4 days or 96 hours or 41·414 gallons per charge of 8·39 tons, or 4,935 gallons for each ton of ore, 658 cubic feet.

At the Cusi mill, Mr. Hofmann states the vats are 12 feet in diameter, taking a charge of about 8 tons of ore. At a filtering rate of 8 inches per hour the volume of outflowing solution amounts to 74·6 cubic feet per hour, or 3,058·6 cubic feet in 41 hours for a charge of 8 tons, or 382·3 cubic feet or 2,867 gallons per ton of ore, employing the Russel process, taking the figures given by Mr. Stetefeldt. Mr. Hofmann, however, states

that he found the average leaching time at the Cusi mill to be 53 hours, which at the filtering rate of 8 inches per hour gives 494·2 cubic feet or 3,706 gallons per ton of ore.

The Stock Solution.

Mr. Hofmann appears to have been the first to introduce the practice of using sodium hyposulphite as a solvent and calcium sulphide as a precipitant. Calcium sulphide contains a considerable amount of (about 6 per cent.) calcium hyposulphite, even if freshly prepared, which in precipitating is introduced into the stock solution. It was therefore supposed that by using calcium sulphide as a precipitant the stock solution would be gradually converted into a solution of calcium hyposulphite, but Mr. Hofmann denies that this is the case, and states that the stock solution remains as sodium hyposulphite. He explains it in this way: At the time when base-metal leaching is usually interrupted, the ore still contains sodium sulphate. Calcium hyposulphite and sodium sulphate form sodium hyposulphite and calcium sulphate, which precipitates

$$CaS_2O_3 + Na_2SO_4 = CaSO_4 + Na_2S_2O_3.$$

If therefore the stock solution after precipitation, containing calcium hyposulphite, is rinsed, and comes in contact with the sodium sulphate contained in the roasted ore, the outflowing solution will have its calcium hyposulphite substituted by sodium hyposulphite, leaving the precipitated gypsum in the ore.

Thus the stock solution, even after continued use, still consists of sodium hyposulphite. It takes longer boiling to manufacture calcium sulphide than to prepare sodium sulphide, but $3\frac{1}{2}$ to 4 hours' boiling is sufficient, and the consumption of steam is small to maintain the solution, once it is boiling, at that temperature. The calcium sulphide appears to have the great advantage that it frees the stock solution from the very injurious presence of sodium sulphate, for if calcium sulphide is brought in contact with sodium sulphate, sodium sulphide is formed, which goes into solution, and the calcium sulphate is precipitated; thus all the sodium sulphate which the stock solution receives from the ore during lixiviation is decomposed, while sodium sulphide if used as a precipitant not only leaves the sodium sulphate undecomposed, but even furnishes an additional supply of this salt, and assists in charging the stock solution with it, diminishing its dissolving energy for silver chloride, and causing quite a large consumption of sodium hyposulphite.

If in preparing calcium sulphide, boiling is not continued beyond 3 or 4 hours and precipitation is properly executed, the original stock

solution can be maintained effective for several years without adding any fresh sodium hyposulphite to it. It will be even found that the solution increases in strength, and in order to keep it at standard strength it requires to be diluted from time to time with water. This cannot be accomplished if sodium sulphide be used. The loss of hyposulphite by decomposition and other causes is in fact more than replaced by the supply derived from the precipitant.

Mr. Hofmann gives a table* showing the tendency of the sodium hyposulphite solution to increase in strength, as above explained, and remarks that it is quite an important financial item whether its standard strength (0·50 per cent.) is kept up by adding water or sodium hyposulphite. At the Cusi mill, he used 765 lbs. of sodium hyposulphite for the preparation of the stock solution, working with it 2,011 tons of ore, and left the solution in perfectly good condition, only a little stronger than when originally prepared, without adding any extra sodium hyposulphite, while Mr. Daggett, who used sodium sulphide, reports a consumption in the Cusi mill of 3 to 7 lbs. of sodium hyposulphite per ton of ore.

Mr. Hofmann states that in working 2,011 tons of Cusi ore containing 45·2 ounces of silver per ton, the cost of calcium sulphide per ton of ore was as follows :—

	Cents.
Sulphur, 3·92 lbs. at 7 cents	27·4
Lime, 8·25 lbs. at 1 cent	8·2
Total	35·6

or 28·7 cents less than the cost of sodium sulphide (reckoned at 64·3 cents) treating an ore containing 35·1 ounces of silver. Reducing the above figures to what they would be in working an ore containing 35·1 ounces of silver per ton, they would stand :—

	Cents.
Sulphur, 3·02 lbs. at 7 cents	21·1
Lime, 6·40 lbs. at 1 cent	6·4
Total	27·5

or 36·8 cents less than the cost of sodium sulphide, which is equal to a saving of 57·2 per cent. Taking into consideration the fact that the use of sodium sulphide causes also a consumption of 3 to 7 lbs. of sodium hyposulphite per ton of ore, it is apparent that sodium sulphide is more expensive, and that calcium sulphide is preferable as a precipitant, regardless of other advantages, for the important reason that it keeps the stock solution free from sodium sulphate.

* *The Engineering and Mining Journal*, New York, vol. xlvii., page 236.

The calcium polysulphide is manufactured at the works, by boiling 2 parts of the fresh lime with 1 part of pulverized sulphur in water for 3 to 4 hours, in deep tanks made of boiler iron, into which steam is directly introduced. The consumption of sulphur is from 2 to 7 lbs. per ton according to the ore.

The Precipitate.

Mr. Hofmann gives the analysis of two different lots of roasted precipitate of del Oro ore:—

	Per Cent.	Per Cent.		Per Cent.	Per Cent.
Gold ...	0·04	0·014	Zinc ...	4·30	13·86
Silver ...	19·00	21·60	Lime ...	3·88	3·62
Lead ...	30·64	21·10	Sulph. acid	6·10	6·18
Copper	11·55	4·44	Sulphur	14·90	19·37
Cadmium	3·45	1·20	Insoluble	5·45	4·96
Iron ...	0·72	2·68			

The two lots of precipitate differ, as will be seen, considerably with regard to lead, copper, cadmium, zinc, and sulphur, due partly to variations in the character of the ore, of which the precipitate is the resulting product, and to a great extent to variations in roasting.

The quantity of sulphur depends on the length of time the precipitate is subjected to roasting in the reverberatory furnace. To avoid loss by volatilization, the precipitate was left in the furnace only until the blue sulphur flame ceased. The percentage of lime (3·88 to 3·62) is in both lots nearly the same, and shows that the value of the precipitate is not depreciated, by using calcium sulphide as a precipitant, enough to make its use objectionable. The dried precipitate contains 47·96 per cent. of sulphur. Experiments to regain the surplus sulphur by boiling the fresh precipitate with caustic soda gave very satisfactory results. They showed that 60 per cent. of the sulphur originally contained in the precipitate can be thus regained and brought into a state in which it can be directly used again as a precipitant.

Using Cupric Chloride for Badly-roasted Charges.

If an insufficiently chloridized ore be treated during base-metal leaching with a dilute solution of cupric chloride it has a very beneficial effect, as stated elsewhere, and Mr. Hofmann, in some instances, obtained a further extraction from the del Oro ore of 34 to 40 per cent. of the silver by its use. 35 lbs. of bluestone and about 70 lbs. of salt, boiled by steam for

about 15 or 20 minutes, gave a sufficient quantity of cupric chloride for a charge of $8\frac{1}{2}$ tons, at a cost of 60 cents per ton of ore:—

	Dollars.
Bluestone, 35 lbs. at 12 cents ...	4·20
Salt, 70 lbs. at 1·7 cents	0·88
Total	5·08, or 0·60 cents per ton.

The cupric chloride is either added to the water contained in the tank, into which the dry and hot ore is dumped, or it is added during base-metal leaching. In the latter case it is better to apply the copper solution after the main portion of the base-metal salts have been leached out, say about 1 hour after commencing base-metal leaching, and to add it gradually in order to penetrate the whole charge. To about 6 or 8 inches of water standing above the ore, one quarter of the prepared copper solution is added, stirred, and allowed to sink through the ore. As soon as the liquid is level with the top of the ore, the outlet under the filter is closed, and again 6 or 8 inches of water is allowed to flow into the vat, to which the second quarter of the copper solution is added; this is repeated a third and fourth time, and the charge is washed in the usual way.

The first method is quicker and less troublesome, but in this case leaching from below is advisable. The solution from the del Oro ore treated in this way left the tank colourless with only a slight reaction for copper, showing that the cupric chloride was decomposed in passing through the ore. Mr. Hofmann believes that if the roasted ore of the Yedras mine were treated in this way, very good results would be obtained, without reducing the fineness of the precipitate as much as is done at present by the use of extra solution.

Cost of Lixiviation.

The works at Parral have two leaching-plants, one in connexion with the modified Howell furnace and the other with the reverberatory furnaces. Separate accounts were kept of each. The following figures refer to the Howell plant. For the consumption of sulphur no separate account could be kept, and the quantity used per ton is therefore calculated on the total amount consumed and the total number of tons of ore leached in both mills. It gives a consumption of 7 lbs. per ton of ore. One man was employed roasting the precipitate obtained from both plants, and one-half of his wages is charged in the statement. A reduction of cost, it is stated, could be effected if the precipitate were boiled with caustic soda, instead of roasting it. The item of management, office, etc., represents one-third

of the cost for 100 tons per day, calculated proportionately on $8\frac{1}{2}$ tons, as explained in the cost of roasting :—

	Dollars.
Labour for charging and discharging	1·00
Two leachers at 1·00 dollar	2·00
One man preparing calcium sulphide	0·75
Sulphur, $59\frac{1}{2}$ lbs. at 6 cents	3·57
Lime, 180 lbs. at 0·5 cent	0·90
One man roasting precipitate at $1·00 (one-half)	0·50
Wood for roasting precipitate	0·50
Management, office, mechanics, assay office	1·78
Oil, light, filter cloth, shovels	1·50
Steam for pump, and sulphide solution were charged to roasting	—
Total cost	12·50

or cost of leaching per ton of ore, 1·47 dollars, Mexican currency.*

Total cost of reduction, not including stamping :—

	Dollars.
Roasting in the reverberatory furnace ...	3·67
Lixiviation	1·47
Mexican dollars	5·14 per ton of ore.

Mr. Hofmann noticed that heavy brown fumes were emitted from the ore in roasting it in the muffle furnace under certain conditions, and was led to suspect the presence of cadmium in the del Oro ore in considerable quantities. Further investigations proved this to be the case. The cadmium leaches out along with the zinc, and as long as there is zinc in the solution, cadmium will be found. The fact that the cadmium is brought into solution by the regular operation of the process for extracting the silver, permits of its extraction (as a bye-product) at very small cost.

The analysis showed that the base-metal solution of the del Oro ore is remarkably free from metals which are precipitable by zinc ; if, therefore, the more concentrated part of the base-solution be conveyed into tanks (like those devised and recommended by Mr. Stetefeldt), for precipitating the copper and silver of the base-metal water with scrap-iron, and metallic zinc is introduced, it will precipitate the cadmium, copper, and silver. The base-metal solution is acid enough, but the addition of some sulphuric acid hastens the process.

It is more profitable to manufacture cadmium sulphide than to produce the metal. The metallic precipitate, after being washed, is boiled with dilute sulphuric acid. Cadmium dissolves, while the copper will remain

* The Mexican dollar was at the time in question, the writer believes, worth, 3s.; at present it is worth 2s. 9d.

as a sediment; and so will lead if it be present. The solution is decanted, filtered, and the cadmium precipitates as cadmium sulphide by sulphuretted hydrogen. Cadmium sulphide is a brilliant and valuable paint. Experiments on a large scale showed that from 2 to 3 lbs. of cadmium sulphide could be precipitated from the base-metal solution derived from a ton of ore.

The best orange-yellow of cadmium is obtained by precipitating with sodium sulphide, but the solution must first be made alkaline with caustic soda.

Mr. F. M. Watson in a letter to *The Engineering and Mining Journal*, New York, of April 8th, 1893,* speaking of the Russel process at Sombrerete, Zacatecas, says :—The ore runs roughly 10 per cent. of blende, 10 per cent. of galena, 30 per cent. of iron pyrites, and the rest mostly quartz containing on an average about 20 per cent. of sulphur. The old reverberatory furnaces used for roasting (taking six months' average) showed a roasting chlorination of 88·5 per cent., but the loss from volatilization was very heavy (the last six months' average being 16·6 per cent., and before it had risen to as high as 24 per cent). By considering the two losses Mr. Watson therefore places their efficiency at 73·8 per cent. The extraction of sulphides during this period was about 73 per cent., and the total cost, including grinding, was about 8 dollars, Mexican currency, per ton, crushing with rolls. He advocates using stamps in place of them. The ordinary Stetefeldt furnace also does not appear to have been more successful in this case than with the San Francisco del Oro ore, but Mr. Watson considers that it might be modified to give good results, and he considers that with a remodelled mill 85 per cent of the raw ore value could be extracted at a cost of not more than 6 dollars, Mexican currency, per ton.

Trough-lixiviation.

This system is a continuous one—a modification of tank-lixiviation—the chemical reactions are the same, but the time of leaching is enormously shortened, and the manipulations are simpler and more labour-saving. It is particularly adapted for large works, and for ore which on account of lead requires a long leaching time.

While the silver from all the other silver-bearing minerals can be easily and quickly extracted as chloride, that portion contained in the galena dissolves very slowly. The larger portion of the silver may be in fact extracted in a few hours, while the remainder will take days. The

* Page 316.

roasted del Oro ore, for example, filters quickly, but if the ore contain 10 to 11 per cent. of lead, the silver-leaching time is 4 days, and if it contain 15 to 17 per cent. of lead the time is increased to 5 or 6 days.

A description of trough-lixiviation is given by Mr. Hofmann in *The Engineering and Mining Journal*, New York, of September 10th, 1887, November 26th, 1887, and March 16th, 1889,* from which these particulars are taken. Silver chloride contained in roasted ore almost instantly dissolves if rapidly brought in contact with a proper volume of moving sodium hyposulphite solution. No more than $\frac{3}{4}$ to $1\frac{1}{2}$ minutes are required, and it is rather the quantity or volume than the concentrated state of the liquid solvent, that produces this effect; it is in fact the principle upon which the process depends.

In the tanks the amount of solution which comes in contact with the ore is regulated by the latter's filtering capacity. The operator can slightly increase the speed of filtering and so increase the quantity of solution used, by producing a vacuum under the filter bottom, but he cannot produce and maintain at will a certain favourable proportion of ore and solvent, which is of such importance for a quick and thorough extraction. If, however, the ore be introduced into a running stream of the solvent, the operator has it in his power to produce and maintain any desired proportion.

Instead of charging the ore in tanks, and permitting the solvent to filter through, it is dissolved outside the tanks in troughs, while the ore is moving in and with the stream of solvent, and the tanks are only used for the separation of the solid from the liquid. The ore from the furnaces is dumped into bins, from which it is mechanically and evenly charged into a perpendicular square tube 12 by 12 and about 2 feet in height, which is crossed by several sprays or sheets of water, closing it near the top and bottom. The dust caused by the contact of hot ore and water is absorbed by the top and bottom sheet of water, and cannot escape, while the steam generated is condensed.

The pulp, when leaving the perpendicular tube, enters a grinding machine of similar construction to the German kegelmühle, an arrangement which answers better than an agitator, which was first proposed for the purpose of mixing the ore with water (to prevent it being carried down the trough in bulk), and to mash and grind up any lumps it contains. The mill discharges the pulp into the base-metal leaching-trough of triangular section, which leads to the settling-tanks.

*Pages 185, 393, and 255 respectively.

By this system, the whole cooling-floor manipulations are avoided, likewise the filling of cars on the cooling-floor and the transportation of the moistened ore to the tanks, also the shovelling out of the tailings from the tanks, which is a great saving of manual labour, without involving much extra machinery. It requires two men per shift to bring the ore from the furnaces to the bins, one man in the base-metal department, and one man and a helper in the silver-leaching department. The stirring of the silver solution in precipitating is done by mechanical stirrers, which do excellent work.

The rapidity of the extraction requires quicker circulation of the solution entailing the use of a larger pump ; but Mr. Hofmann considers the extra expense of this a very small drawback. The hot ore warms the water for base-metal leaching, without incurring extra expense or labour.

The proportion of water depends on the amount of base-metal chlorides contained in the ore, and has to be so regulated that the resulting base-metal solution is too dilute to dissolve silver chloride. This is easily arranged, by gradually increasing the stream of water while maintaining the same supply of ore, and testing the resulting solution for silver.

The length of the trough depends on the character of the ore, but 150 feet from the mill to the first tank will, in most cases, be sufficient. The inclination should not be less than $\frac{3}{4}$ inch to the foot. The trough can be arranged in zigzag, but has to lead over all the tanks.

In the line of the trough above each tank there is a square box, 14 by 14 by 10 inches, the bottom of which is provided with a plug-hole in order to permit of any desired tank being charged.

The tanks have in the centre of the bottom a sluice-hole 6 inches in diameter, to which is attached to the outside by means of a flange a short piece of cast-iron pipe of about the same diameter. This pipe is like a gas-pipe elbow, and can be tightly closed by pressing a rubber gasket against the outside flange. This valve is worked from a platform alongside the tanks. This pipe must be well coated with asphaltum varnish. Around the sluice or discharge-hole the filter is arranged in funnel-shape, having an inclination of $\frac{3}{4}$ inch to the foot. The filter-cloth must be well fastened round the outer and inner circle. The central position of the discharge-opening and funnel-shape of the filter-bottom permit of quick and perfect sluicing. A tank with a level filter-bottom, if large, can be sluiced clean through a side gate. The space below the filter is provided with the usual outlet-pipe. Close under the filter is inserted from outside a $\frac{3}{4}$ inch pipe which, connected with 1 inch hose, reaches the rim of the tank for the escape of the air. Before the tank is

used for operation, a wired rubber-hose is connected with both the water and the solution-pipes, and so placed from above that the lower or outlet end enters the discharge-pipe through the central hole in the bottom. The object of this hose is to inject a stream of solution at the bottom of the ore (in the outlet-pipe), when the tank is to be sluiced out for silver leaching.

The stream undermines the tightly-packed ore which gradually caves in, until a funnel-shaped opening is made through its depths. Then one or more streams are allowed to play on top of the ore until it is all sluiced out. The hose in the discharge-pipe—which must be stiff in order not to be flattened—is allowed to remain, in order to prevent the discharge-hole from being clogged by a too sudden rush of ore. Before starting the operation the discharge-pipe is filled with water through the central hose, the lower end being covered, in order to keep it filled with water to prevent obstruction from the ore.

The tanks are placed on the same level in two rows close together, and are connected by pipes in such a way as to permit of any desired tank being disconnected without disturbing the communications of the others. The connecting-pipes are also on a level, placed a few inches below the rim of the tanks, and well coated with asphaltum varnish. Their diameter depends on the capacity of the works. Each tank is provided with an outlet (on the level of the connecting-pipes), which discharges below the bottom into the base-metal trough, the latter being connected with the outlet-hose from under the filter-bottom. The connecting-pipes and upper outlets can be closed with wooden plugs from the inside of the tanks.

Base-metal Leaching.—At starting, the stream of roasted ore and water, after having passed through the whole length of the base-metal leach trough, is allowed to enter the tank by opening the hole in the bottom of the small square box which intersects the trough above the tank. The ore will gradually fill the first tank, while the base-metal solution after reaching the level of the connecting-pipe will flow into the next tank, and when this is filled, into the next tank, and so on through all the tanks until it finds the outlet of the last tank, through which it will discharge into the base-metal trough. The motion of the solution in each tank is so slow that, by the time it reaches the last outlet, it is clear. A board, placed edgeways a few inches below the surface and across the tank, will prevent the formation of a diametrical stream from one pipe to another, and greatly assist in clearing the solution. If the proper proportions of water and ore have been used, the base-metal

solution will not contain any silver, and can be allowed to run to waste. When the first tank is filled with ore, the connecting-pipe is closed, and stream of pulp transferred into the second tank, all other connexions remaining unchanged. The outlet-hose below the filter-bottom of the first tank is now opened, and the solution still contained in it is allowed to drain off. As soon as it disappears below the surface of the ore some clear water is added to force out the remainder of the base-metal solution. The first tank is now ready for silver leaching.

Silver Leaching.—In their passage through the trough from the mill to the tanks, all the soluble base-metal chlorides have dissolved, and the charge of the first tank having been treated, as above described, is now ready for silver leaching. The rubber gasket of the discharge-pipe is pulled back, and sodium-hyposulphite solution is turned on through the central hole, and the whole charge sluiced out. Ore and hyposulphite solution discharge into a trough under the first and last tanks, which are connected with the silver-leach trough. At first, before other streams of hyposulphite solution can be played on the surface of the ore, the pulp is diluted by an extra stream in the silver-leach trough. Ore and solution now pass through a trough, not less than 150 feet long, to a similarly arranged set of tanks.

When the pulp reaches the first tank, the solution has dissolved all the silver chloride, and the sand drops as tailings into the tank. The silver solution has also to pass through all the tanks and leaves the last one clear. It is conveyed to the precipitation-tanks. When one tank is filled with tailings, it is disconnected from the others, and the pulp allowed to flow into the next one. The outlet-pipe under the filter-bottom is opened, the remaining silver solution allowed to drain off, and the part retained by absorption displaced by water. Where water is scarce, the base-metal solution can be accumulated in the outside storage-tank and used for sluicing the tailings. The proportion of solution and ore to be used depends on the nature of the ore, and has to be ascertained for each kind. Ore containing lead requires most solution. It is best to determine the proportion in weights, as it makes it easy to calculate the required capacity of the different tanks, pipes, and pumps, for a given capacity of mill.

Mr. Hofmann's tests, he states, show that the minimum quantity of solution required is 3 to 5 times the weight of the ore, and the maximum 18 to 20 times. The weight of the solution is, for convenience, taken as equal to water, 1 cubic foot = 62·5 lbs. In most cases 10 parts of solution to 1 of ore will be sufficient. Assuming this proportion, and

taking 1 cubic foot solution as 7·5 gallons, and 1 gallon as 8·33 lbs., we find that in order to lixiviate 40 tons per day, it will require 3,200 cubic feet or 24,000 gallons of solution to be daily pumped to the upper reservoir and circulated. This, theoretically, would require a pumping capacity of 16·66 gallons per minute, and precipitating facilities for 1,000 gallons per hour. By erecting a temporary trough of not less than 150 feet in length, and a mixing-box at the upper end, the required proportions of solution and ore can readily be determined by a series of experiments. The samples have to be taken at the lowest end of the trough, while the pulp is dropping into the receiving-tank. The vessel with which the sample is caught must be large enough to receive the whole stream during the time the sample is caught. Sufficient time must be allowed for the ore to settle. The clear solution is carefully decanted, and the sediment placed on a filter, washed well with water, dried, and assayed. Before subjecting it to the solution-test, the ore has, as a matter of course, to be treated with water to remove the base-metal chlorides.

Mr. Hofmann states the advantages of trough-lixiviation as follows:—

At first sight it would appear that very dilute silver solutions are obtained, and have to be treated in the precipitation-tanks. This, however, is not the case. The resulting silver solution maintains a uniform strength in silver and other soluble metal salts, and is of about the same strength as the whole filtrate of one charge in tank-leaching would be, if accumulated in one precipitation-tank. In tank-lixiviation very strong solutions are formed at the beginning, and very dilute ones towards the end, and as it frequently happens that a tank-full of such very dilute solutions has to be precipitated by itself; a uniform strength is in fact by far preferable.

This method of lixiviation allows the operator to bring the ore in sudden contact with any desired quantity of the solvent, and offers the means in this way, of overcoming some very annoying chemical and mechanical difficulties encountered in other lixiviation processes. Lead sulphate reduces the dissolving energy of sodium hyposulphite for silver, and this is why the lixiviation of lead-bearing ores is so exceedingly slow and requires such an extensive plant.

In the ordinary lixiviation, the solution becomes more saturated with lead sulphate as it descends through the ore and loses proportionately its dissolving energy. As the solubility of the lead sulphate increases with the concentration of the solution, a stronger solution does not hasten the process; but if the ore be brought rapidly into contact with a large volume of hyposulphite solution, the latter retains enough of its dissolving

energy to produce a quick extraction. The presence of lead sulphate therefore does not retard trough-lixiviation, it merely entails the use of larger quantities of solvent. The rapidity with which the silver dissolves, also materially lessens the unfavourable influence of caustic lime. The possibility of bringing the ore into contact with any desired quantity of the solvent has the further advantage of preventing the base-metal solution from dissolving silver. The resulting solution need only be sufficiently dilute. It does not take much more water than in the ordinary leaching, when the bulk of the silver is dissolved by the base-metal chlorides, while the solution is very concentrated. Shortly after the beginning, no more silver is dissolved; if, therefore, the charge of ore could have been brought at once into contact with the whole quantity of water used in washing, Mr. Hofmann thinks much less silver would go into solution. Clayey ores and flue-dust may be successfully treated by trough-lixiviation regardless of their filtering property.

In ordinary leaching, each tank-charge is in a different stage of the process, and this necessitates great care and attention and keeping a separate record of each tank. In trough-lixiviation, the operation being continuous, attention need only be concentrated on one tank in each department to prevent it from being overcharged. Although the process is divided into two departments (the base-metal and silver leaching) the manipulation is materially simplified, the tanks being charged automatically by the stream of the respective solutions, and the costly handling in cars is obviated. Leaching works built on this plan will require more grade, and it will be preferable to have the roasting and base-metal leaching done in the main building; and the silver settling and precipitation-tanks, with all the other apparatus required for the final treatment of the precipitate, in a separate department lower down; the two being connected by the silver-leach trough.

The principle of this method can be tested in the laboratory by introducing 20 grammes of roasted ore, which has previously been washed into a graduated cylinder of 1,000 cubic centimetres, in which is contained 200 cubic centimetres of sodium hyposulphite solution. The top of the cylinder has to be tightly closed with the palm of the hand, and the glass brought into a horizontal position and oscillated, in order to make the ore and solution pass quickly from one end to the other to imitate the current in a trough. After the oscillating motion has been continued for about 2 minutes the contents of the cylinder are emptied into a filter, and washed with water to displace the silver solution from the sand and filter-paper.

It will be found that all the extractable silver has been dissolved, and the residues will correspond with those obtained in the regular chlorination assay. If the result of a first trial be not satisfactory, it is due to a fault in the quantity of solution, more of which must be used in the next test, and so on, till the results are accurate and the required proportions of solution and ore are ascertained.

Trough-lixiviation was first used in the mill of the North Mexican Mining Co., Cusihuriachic, Chihuahua, Mexico. This mill was originally arranged for tank-lixiviation, but the ore hardened like cement, and did not permit any solution to pass through. Trough-lixiviation, however, enabled the ore to be worked rapidly, and the resulting tailings agreed with the chlorination test.

Mr. Hofmann gives a number of experiments and details with regard to trough-lixiviation in *The Engineering and Mining Journal*, New York, of March 16th, 1889,* and in the *Transactions* of the *American Institution of Mining Engineers*, February, 1888.† These cannot, however, be reproduced here in detail. As the saving in time is, however, one of the chief claims of the process, the writer may instance some comparative figures Mr. Hofmann gives, comparing it with tank-lixiviation treating del Oro ore :—.

Time taken in tank-lixiviation :—

	Hours.
Charging	3
Base-metal leaching	8
Expelling the water by solution	1
Silver leaching	96
Expelling the solution by water	1½
Total	109½

Time taken in trough-lixiviation :—

	Hrs.	Min.
Base-metal leaching and filling the tank	3	6
To drain the wash-water from the top of ore ...	0	34
To expel the base-metal solution by water ...	3	25
To expel the water by hyposulphite solution ...	1	20
Silver leaching (sluicing with solution)	3	36
Draining solution from top of ore	0	34
Expelling the solution by water	2	30
Total	15	5

Treating the del Oro ore by trough-lixiviation, Mr. Hofmann found in silver leaching that, using a solution with a strength of 0·50 ounce per ton, the best results were obtained with the proportion of 1 of ore to 3·4 of solution or 108·8 cubic feet or 816 gallons of solution in circulation

* Page 255. † Page 662.

per ton of ore treated. But in trough-lixiviation, 658 cubic feet, or 4,935 gallons of solution were required per ton. The proportion of 1 to 3·4 gave tailings carrying only 3·59 ounces per ton. In evidence of the statement that by producing a sufficiently dilute base-metal solution it will not contain any silver and may be allowed to run to waste, Mr. Hofmann cites the following test :—One litre of the 702 gallons of base-metal solution was precipitated with calcium sulphide. The precipitate, after fluxing and treating like an ordinary ore assay, returned no more than 0·0002 gramme of fine silver, 702 grammes will therefore contain 0·532 gramme, which represented the total amount of silver dissolved from 8·39 tons of ore, or 0·06 gramme or 0·002 ounce of silver per ton, which is practically nothing.

Other points mentioned are, while in tank-lixiviation the rate of filtration was 8½ inches per hour, in trough-lixiviation it was 12 inches per hour. The number of tanks required in trough-lixiviation is much less than in tank-lixiviation, and while the roasted sulphides obtained from the trough-process contained 20·9 per cent. of fine silver, those obtained from the tank-process during the same week and from the same lot of ore only contained 17 per cent. fine silver.

Some of the most interesting figures may be summed up as follows :—

	In Tank-lixiviation.	In Trough-lixiviation.
Quantity of water required for base-metal leaching, including sluicing, per ton	943 gals.	1,129 gals.
Quantity of hyposulphite solution which has to circulate for each ton of ore	4,935 „	816 „
Time required to treat one tank-charge of ore ...	109 h. 30 m.	15 h. 5 m.
Total quantity of water required for 100 tons of ore per day	94,300 gals.	112,900 gals.
Total quantity of hyposulphite solution per day to work 100 tons	493,500 „	81,600 „
Loss of silver in base-metal leaching per ton of ore...	0·25 oz.	trace.

Extraction of silver in both methods the same.

In connexion with the Hofmann gold-and-silver chlorination process, which has been mentioned in the earlier part of this paper, it is to be noted that after the silver has been extracted, the solution of hyposulphite used for leaching is allowed to run out till it disappears under the surface of the ore, when clear water is introduced, in order to displace the solution. The desilverized ore must then be removed from the tank to a dry-kiln, where it is left till the surplus water has evaporated. After this it is charged back into the tank still moist.

This second handling and drying cannot be dispensed with, as the ore after leaching is too wet to permit of the free passage of chlorine. If the

ore be cupriferous, much copper will be carried out with the gold solution (after it has been chlorinated), colouring it green.

An application of the leaching process to the pan-amalgamation of very base ores, which has been invented by Mr. Kustel, deserves notice. He states that it is applicable to silver ores, like those of Flint, Idaho, containing base-metals, and also to auriferous copper ores, which by their nature require roasting, as the amalgamation of gold is very much obstructed by the presence of copper salts. If there be soluble chloride of silver in the roasted ore, and besides this, soluble chlorides of copper, lead, antimony, and zinc, they will all, as a matter of course, be decomposed and amalgamated.

All take part in consuming and flouring the quicksilver and in destroying the pan, such a combination as the above hindering the amalgamation of the silver and gold. The base-metal chlorides being soluble in water, while the chloride of silver is not, it is a simple expedient to dissolve out these salts by leaching with hot water, and thus remove them from the ore, prior to amalgamation. The ore being thus divested of its rebellious features, gives excellent results in the pans.

Lixiviation *versus* Amalgamation.

In comparing the Russel process with amalgamation, Mr. Stetefeldt* enumerates the principal points in favour of lixiviation as follows :—

1. In amalgamation, the coarseness of crushing, without considering the question of roasting, is limited by the capacity of the settler to work off coarse sands without loss of quicksilver. In lixiviation pulverizing as coarse as possible is desirable.

   The limit of coarseness depends on the character of the ore, and principally upon the manner in which the silver-bearing minerals are distributed in the gangue.
2. The original cost of the lixiviation-plant is much lower than that of pans and settlers, and a further saving is effected by a reduction in the size of engines and boilers.
3. In amalgamation the pans and settlers consume not less than $1\frac{1}{3}$ horse-power per ton of ore. The power for pumping solutions, etc., in the lixiviation process, is merely nominal.
4. In large mills the quantity of quicksilver in solution represents a capital of 30,000 to 40,000 dollars (£6,250 to £8,333 6s. 8d.), while the stock of chemicals required for lixiviation costs less than one-tenth of this amount.

* *The Lixiviation of Silver Ores*, page 4.

5. With the Russel improvements the percentage of silver extracted by lixiviation is in most cases higher than by amalgamation.
6. Lixiviation by the Russel process requires a less careful chloridizing-roasting, and in consequence a lower percentage of salt may be used in roasting.
7. Ores that can be successfully treated by raw amalgamation give often better results by lixiviation with extra solution.
8. The value of the quicksilver lost, and cost in wear and tear of the pans and settlers amounts to more than that of the chemicals consumed in the lixiviation process.
9. The lixiviation process permits of the extraction of copper and lead as valuable bye-products.
10. Amalgamation is invariably injurious to the labourers' health.
11. Where gold-bearing silver-ores have been roasted with salt, lixiviation extracts in many cases more gold than amalgamation.

The disadvantages of lixiviation as compared with amalgamation are :—

1. Lixiviation requires more chemical knowledge and a more careful supervision of the operations.
2. The handling of large volumes of solutions is a disadvantage common to all humid processes.
3. There is more danger of losing silver by careless manipulation, and by leakage of badly constructed plant.
4. In the lixiviation process the precious metals are obtained in the form of sulphides. The conversion of the latter into bullion, requires more skill and is more expensive than the handling of amalgam.

The chemistry of the process has been most ably discussed and dealt with by Mr. Stetefeldt and Mr. Ellsworth Daggett,* and treated of in the *Report of the Californian State Mineralogist*, 1888, which contains an independent article upon the hydro-metallurgy of silver, as well as a critical review of the Russel process by Mr. C. H. Aaron.†

The writer gathers from these various sources that it is with roasted ores which have an alkaline reaction caused by the presence of caustic lime (as, for example, the alkaline arsenical ores of Las Yedras, Sinaloa, Mexico,) that the Russel process, as compared with the ordinary Van Patera process (leaving the Kiss process out of the question), has

* *Trans. Am. Inst. Min. Eng.*, vol. xvi., page 362. † Page 832.

achieved its greatest success, but it is possible that it may meet with a formidable rival in this province in the new departure of pyritic smelting.*

Looking at a number of examples that are given it would appear that the extraction in the mill by the Patera process shows a difference in certain instances, in favour of the Russel process, of 22 to 30 per cent. of the silver extracted.

A table given by Mr. C. H. Aaron also shows the results of a series of competitive mill-runs, extending over eight months at the Ontario Mill (on Ontario ore), in which the percentage extracted by the Russel process varied from 84·7 to 93·9 per cent. (or on the average about 91·17 per cent.) as against 65·3 to 84·9 per cent. (or on the average 77·4 per cent.) extracted by amalgamation, treating ore which ran about 47 ozs. per ton roasted. The tailings from the amalgamation still carried 10·76 ozs. per ton on the average, whilst the lixiviation-tailings only ran 4·3 ozs.

Experiments at the same works from November, 1887, to January, 1888, on an ore which ran 43·76 ozs. in silver, showed that 82·1 per cent. was extracted by amalgamation, as against 91·5 per cent. by the Russel process.

On the whole, however, taking the results of five different mills it would seem that whilst amalgamation gave an average of 80·7 per cent., the Russel process averaged 89·4 per cent., or 8·7 per cent. above the results of amalgamation, and 4·8 per cent. better than could be extracted with ordinary solution making the latter tests in the laboratory.

It is therefore to be inferred that, in the case of certain ores which have been specified, the Russel process extracts more silver than amalgamation, but it is to be also noted that in the case of four of the mills taken as examples, the percentage extracted by amalgamation is certainly 10 per cent. less than in many mills where roasting-milling is practised, and therefore cannot be altogether regarded as typical of the capabilities of the latter process in certain cases.

The ore, for example, of the Ontario mine is very base, containing, as before pointed out, zinc, lead, and silver sulphides, as well as chlorides, in a quartz gangue, and therefore the comparative success of the process in this instance, even from a purely chemical standpoint, scarcely foreshadows it as a universal fact.

Next, as regards capital outlay, Mr. Stetefeldt gives the cost of a plant for dry-crushing and roasting 80 tons a day as follows :—

* It is to be regretted that more details of the cost and working of this process are not at present available.

	Weight in Lbs	Dollars.	£	s.	d.
Buck-eye engine, 72 to 121 horse-power	15,000 ...	1,775·00			
Knowles, feed-pump, No. 3	640 ...	275·00			
Heater	2,000 ...	175·00			
Air-pump and condenser		450·00			
Two Krom rock-breakers	12,000 ...	1,500·00			
Three double-shelf dry kilns	70,000 ...	2,700·00			
Two sets Krom 26 inches rolls	29,000 ...	4,500·00			
Two Krom screens	1,200 ...	800·00			
Stetefeldt furnace	49,000 ...	3,000·00			
Hoist, with cage and safety-catches ...	2,000 ...	600·00			
Shaftings, bearings, pulleys, wire-rope transmission	18,000 ...	1,300·00			
Electric light plant, with separate engine for dynamos	... ...	1,700·00			
Belting	... ...	1,000·00			
Elevators, conveyors, feeders, hoppers, cone-pulley	... ...	1,000·00			
		20,775·00	= 4,328	2	6
Add the cost of four 40 horse-power high-pressure boilers	... ...	...	...		
Add the cost of the lixiviation-plant proper,* with a capacity of 80 to 170 tons of roasted and raw ore, if the tailings are removed by sluicing; 70 to 140 tons, if the tails are to be shovelled	... ...	6,772·70	... 1,310	19	7
Add the cost of painting vats and tanks with three coats of white paint, say, 746 lbs. of white lead and 41·7 gals. oil, occupying one man 34 days to lay on	... ...	...	... ...		
Add the cost of plant for refining sulphides by the humid process estimated (erected) at	... ...	5,000·00	... 1,041	13	4
Add the cost of grading and foundation to the mill structure and plant it contains (which it is only possible to figure in accurately by knowing the exact quantities of material and the time of skilled and ordinary workmen employed on the job)	... ...	...	... ...		

* This includes finished lumber and hoops for 6 lixiviation-tanks, 3 storage-tanks, and 6 precipitating-tanks, 1 sulphide storage-tank, 2 solution-sumps, cast and wrought-iron fixtures for vats and tanks, pipes, valves, steam-hose, 1 sodium sulphide mixing and 2 storage-tanks, 8 Korting ejectors, No. 4 Knowles fire-pump, (size A), for sluicing tails; Knowle plunger-pump for pumping solutions, 1 Johnson 18 inches filter-press, 1 Johnson pressure-tank, and a Knowles feed-pump, No. 2, for boiler.

Add finally freight and transport charges, depending on the locality, and it will be found that the total cost is not so far short of a pan-amalgamation plant of the same capacity as might be supposed ... ...

Now, let us glance at the working expenses of running a lixiviation-plant which are given by Mr. Stetefeldt for a mill of 80 tons capacity per day :—

	Cords of Wood.	Tons of Coal.
Fuel, engine and boilers*	5 to 6 ...	$2\frac{1}{2}$ to 3
Boiler lixiviating-building	3 ...	$1\frac{1}{2}$
Dry kiln ore and salt	$3\frac{1}{2}$ ...	$1\frac{3}{4}$
Stetefeldt furnace, depending on quality of wood and character of ore	$6\frac{1}{2}$ to $8\frac{1}{2}$	
Sulphide refinery for roasting, refining, and melting bars in a reverberatory furnace	1	
Total ...	19 to 22	

The cost of labour per day would be as follows :—

	No. of Workmen.		Hours of Work.
Engine and boilers ...	2	engineers	12
,, ,, ...	2	firemen	12
Ore-house	2	rock-breakers	10
,, ...	2	wheeling ore	10
Dry kilns	4	bringing ore from ore-house and charging kilns	12
,, ...	2	discharging kilns	12
,, ...	2	firemen also attending salt-kilns and assisting rolls	12
Rolls	2	machinists	12
Stetefeldt furnace ...	3	firemen	8
Cooling-floor	12	discharging, cooling, and charging vats ...	8
Lixiviations	2	lixiviators	12
,, ...	3	helpers	12
,, ...	2	precipitators	12
,, ...	2	firemen on boilers, pumps, and pipes	12
,, ...	2	handling sulphides	12
Refinery	2	roasters and refiners	10
Mill	1	night foreman	12
,, ...	1	machinist	10
,, ...	1	carpenter	10
,, ...	1	blacksmith	10
,, ...	4	general helps	12
,, ...	1	assayers' help	12
,, ...	1	blacksmith's help	10
,, ...	1	electrician	—
,, ...	1	bookkeeper	—
,, ...	1	assayer	—
,, ...	1	chief metallurgist	—
,, ...	$\frac{1}{2}$	chief mining engineer	—
,, ...	2	men and team hauling wood	—
Total ...	$62\frac{1}{2}$		

* It is assumed that 1 cord of wood is equal in effect to 1,000 lbs. of coal.

The daily consumption of chemicals is as follows:—

	Per Ton of Ore. Lbs.
Copper sulphate	1·8 to 9·6
Sodium hyposulphite	1·5 to 7
Caustic soda	1·4 to 7·75
Sulphur	0·9 to 5
Sulphuric acid	0·25 to 1·80

The water required is from 9 to 55 cubic feet per ton of ore, exclusive of that used for sluicing tailings.

The cost of wear and tear to cover all ordinary breakages, and wear and tear of machinery, boilers, screens, dry kilns, furnaces, lixiviation-plant, and refinery, including lubricants, electric light, and sundries, is about £7 5s. 10d. per day. For plants of small capacity the expenses per ton of ore, for labour, are, of course, materially increased. To the above must be added insurance, taxes, interest, legal expenses, etc., and amortization on the capital invested in the mill.

The total expenses at Cusi, Mexico, including refining, are said to have been £2 10s. 4d. per ton, using the Russel process.

For roasted ore at the Sierra Grande mill, Lake Valley, the total mill expenses for 60 tons per day were estimated by the general manager, Mr. Hadley, at 19s. 4d. per ton.

At Parral, Mexico, the total expenses for the treatment of tailings (from ore which had originally been roasted and lixiviated by the ordinary process) were 8s. 9d. per ton for 40 tons per day. Treating roasted ores at the rate of 10 tons per day the total expenses were £1 18s. 1½d. per ton.

The cost of raw-leaching tailings at Silver Reef, Utah, treating 25 tons per day (the tailings assaying 6½ to 9½ ozs. of silver but no gold), is given by Mr. Egleston* as 6s. 10½d. per ton, the percentage extracted averaging 55 to 60 per cent. The presence of copper carbonate in these ores caused the sulphides to have a low percentage of silver (3,560 ozs. per ton) and a high tenour of copper.

Treating tailings from amalgamation-works at Silver City, concentrated up to 30 ozs. per ton silver, it is asserted that the Van Patera process only extracted 38 per cent. against 72·4 obtained by the Russel process.

Mr. Stetefeldt states that in a well-constructed lixiviation-mill the total expenses for treating 75 tons of raw ore per day should not exceed 12s. 6d. per ton, and under favourable circumstances should fall as low as 10s. 5d. per ton, particularly if the crushing were done by rolls instead of stamps.

* *Metallurgy of Silver*, page 532.

The chemicals required to be carried in stock for a lixiviation-plant of 80 tons daily capacity are estimated in an assumed case as amounting to about £878 0s. 2d., to last for 60 days, including sodium hyposulphite in stock solution.

Mr. Stetefeldt, page 213, says: "The Ontario mill expenses are about £2 14s. 2d.* per ton, those for lixiviation would be about £1 0s. 10d. to £1 5s. less, or a difference in favour of the Russel process of £1 0s. 10d. to £1 5s. in expenses, which, together with the additional extraction of 17s. 1½d., would make a total net difference of £1 17s. 6d. to £2 1s. 8d. per ton in favour of lixiviation."

As already remarked, however, in this the favourite instance taken for comparison of the two processes, the mill extraction by amalgamation is not nearly so high as it might be, and in the same way the cost, is nearly 12s. 6d. above the average of many amalgamation-works. Compare, for example, the expenses of mill B of the Granite Mountain Co. (43 stamps) for the year ending July 3rd, 1891, and it will be found that the milling charges did not exceed £2 4s. 8d. per dry ton, treating 19,463 tons dry, whilst in the case of the Elkhorn mill, which treated 11,645 tons, the cost was only £1 18s. 5¼d.

If one compares the amount of labour required in a lixiviation and an ordinary amalgamation-mill in places where labour is dear, save in some exceptional cases, lixiviation must be the more costly process of the two, and it is in fact in countries like Mexico, where the conditions in this and other respects are favourable that it will find its widest adoption, until the labour costs common to all processes of the kind can be reduced considerably.

In confirmation of this statement the writer may cite the Geddes and Bertrand mill, Nevada† (an ordinary lixiviation-plant dealing with 50 to 60 tons per day), the staff of which, when running, consisted of 60 men, whose wages amounted to £30 6s. 3d. per day.‡ The cost of lixiviation milling (using rolls) came to £1 7s. 1d. per ton, and 13s. 8½d to £1 0s. 2d. in silver was still left in the tub-tailings.

Whilst leaching is doubtless applicable to certain ores in certain

* Mr. Egleston gives the actual running expenses, treating Ontario ore by amalgamation on a production of 50 tons per day as £3 2s. 7d., agreeing with Mr. Rothwell's figures. These works seem in fact to have employed an exceptionally large staff for a 40 stamp mill, viz., 66 to 72 men.

† The failure of amalgamation appears to have been due to the presence of antimoniate of lead in the ore.

‡ Egleston, "Leaching Gold and Silver Ores in the West." *Trans. Am. Inst. Min. Eng.*, vol. xii., page 40.

localities, it is always a process which involves highly skilled superintendence and chemical supervision, otherwise very serious losses may be incurred in roasting, leaching, and precipitating the metals, and in this way alone the possible profit to be gained by employing it may easily be converted into a positive loss.

Treatment of the Sulphides obtained from the Russel Process.

One of the great drawbacks of the Russel process which has been referred to, and which is common to other lixiviation processes yielding a base precipitate, viz., the treatment of the sulphides of lead, copper, iron, silver, and gold, produced by precipitation with alkaline sulphides, is now believed to have been overcome by a process invented by Mr. Cabell Whitehead. This new mode of treatment, it is stated, can be effected at a cost of $1\frac{1}{4}$ cents per ounce of contained silver, at the same time avoiding the frequent loss of silver that other methods entailed.

Refining on a cupellation hearth (the old method which is still in use in several mills) has the disadvantage of causing large losses of silver by volatilization in the previous roasting, and the locking up and eventual loss of a portion of the silver in slags, cupel-bottoms, and matte.

From 80 per cent. to 90 per cent. only of the silver charged was obtained in the shape of bars. The rest would be in greyish-black copper-lead silver matte, which would be formed early in the operation in the pasty slags which it was impossible to get rid of, and again the precious metals had a most pernicious faculty of sinking, not only into the cupel-bottoms, but below even into the iron-pan and into the surrounding brickwork. All these products had to be treated again, or shipped direct to a smelter. If they were shipped differences in assays, and possible losses were bound to occur, as the products contained shot silver. If the sulphides were charged upon the bath without undergoing a previous roasting under the mistaken idea of preventing the loss of silver by volatilization, the same troubles occurred in even a more pronounced degree. If the company shipped its sulphides direct to smelting or refining works, it avoided the great losses of this crude process, but incurred heavy expenses and discounts, amounting to as much as 10 per cent. on an average, in the case of a Mexican mine.

At present interest centres on two new processes, the one above alluded to, invented by Mr. Cabell Whitehead. This process is confined to the treatment of the sulphides as they are found in the precipitation tubs of a lixiviation plant. Its details have not yet been made public, but it is

stated that its success has been experimentally demonstrated, and the refining department of the Marsac mill of the Daly Mining Co., has been remodelled with a view to its adoption.

The other process aims to get at the root of the difficulty by precipitating electrolytically on zinc plates. Its success, whilst doubtful, must yet be proven before it is entitled to serious consideration.*

Examples of the Russel Process.

The careful preparation of the ore by a thorough chloridizing-roasting appears to be one of the chief points on which the success of the Russel process, in such cases as it may be applicable, turns. It does not appear to be adapted for the treatment of ores containing metallic silver, which is comparatively insoluble in cuprous hyposulphite. An account of the results of the process at two of the works where it has achieved its most successful results may be of interest.

For a description of its working at Las Yedras, Sinaloa, Mexico, the writer is indebted to an article in *The Engineering and Mining Journal*, New York, dated January 14th, 1893,† contributed by Mr. R. F. Letts. The Yedras mine of the Anglo-Mexican Mining Company is situated in the north-eastern corner of Sinaloa.

A 40 stamp mill and a lixiviation-plant to use the Patera or Kiss processes was erected in 1882. Poor results were obtained, and the Brückner furnaces which had been introduced were abandoned for the cruder but more satisfactory reverberatory, as the latter did not ball or agglomerate the roasted ore. It is estimated that the Patera process did not save over 65 per cent. of the silver in the roasted ore.

The following are two analyses of Yedras ore, representing the averages of the ore treated at different periods:—

	No. 1.		No. 2.
Carbonate of lime	53·78	...	46·50
Silica	15·13	...	25·00
Iron	17·33	...	9·80
Sulphur	13·31	...	12·50
Arsenic	9·82	...	2·50
Zinc	4·92	...	—
Lead...	1·78	...	—
Magnesia	2·58	...	—
Alumina	1·35	...	—

No. 1 is an analysis of the average battery sample for one month. The composition of the ore varies greatly. A couple of months later, analysis

* *The Engineering and Mining Journal*, New York, February 25th, 1893, page 169.

† Page 34.

No. 2 was made; the battery samples for several weeks showed 4 per cent. of zinc, and two months later contained a large percentage of antimony.

The results by using the Patera process were as follows:—Assay of ore, 60·67 ounces per ton; extraction in assay office, 72·09 per cent.; extraction in mill, 67·12 per cent.; total leaching time, 92 hours.

By the Russel process the results were as follows:—Assay of ore, 55·3 ounces per ton; extracted by old process in assay office, 69·94 per cent.; extracted by Russel process in assay office, 83·62 per cent.: extracted by Russel process in mill, 82·44 per cent.; leaching time, 76 hours.

It will thus be seen that the extraction by the Russel process was 15·32 per cent. higher than by the old process, and that the leaching time was 16 hours shorter. The chemicals consumed per ton of ore were as follows:—

	Old Process. Lbs.	Russel Process. Lbs.
Lime	9·7	—
Sulphur	4·7	3·6
Hyposulphite of soda	—	1·4
Bluestone	—	9·6
Caustic soda	—	5·5
Total	14·4	20·1
Cost per ton ...	$0·52 = 2s. 2d.	$2·20 = 9s. 2d.

Since these runs were made, the consumption of chemicals has been reduced to 9·14 lbs. per ton of ore and the cost to 0·82 dollars.

Owing to the great distance from the railroad, the price of chemicals in Yedras is of course considerably greater than at most places in the United States or at many localities in Mexico. The average cost of chemicals per lb. at Yedras for the last three years is as follows:—Hyposulphite of soda, 8 cents; bluestone, 10 cents; caustic soda, 9·1 cents; and sulphur, 7·1 cents. No soda ash (sodium carbonate) is used at Yedras as there is usually no lead in the ore. The total cost of all chemicals at Yedras in 1890 was 3·6 cents per ounce of silver produced. Of the copper used in the form of bluestone about 50 per cent. remains in the ore.

During the past five years the comparative efficiency of the two processes at the Yedras mill has been tested four times. The duration of each of these tests was from one to three months. Two methods were pursued in making these comparative runs. One was to divide the roasted ore equally between the two processes, running one-half the ore vats and precipitating-tanks on the old process and the other half by the Russel process, the products being kept entirely separate, and the tailings from each process thrown out.

By this method, however, there is a loss of 8 or 10 ounces per ton on all ore treated by the old leaching process, as that amount, which might be extracted by the Russel process if it were used on the same charges after the old process, remains in the tailings.

In the other method all the charges are treated first by the old process, that is, by the simple hyposulphite solution, until no more silver can be extracted, the sulphides being precipitated by themselves and kept separate. Then these same charges of ore are treated by the Russel process, that is, by cuprous hyposulphite or extra solution. The precipitates from this solution are likewise kept separate. In this way a comparison between the two processes is made without any loss, each charge of ore having the benefit of being treated by both processes before it is thrown out. In fact, this is the way all the ore is treated at Yedras, all charges being first treated by the old method, and then by the Russel process.

The first of the two comparative runs was made by Mr. Letts between the two processes in September, 1890: the test lasting a month.

An extract from his report to the Anglo-Mexican Company runs as follows:—"Our intention was to give the old process every possible show. Great care was taken to keep the precipitates separate, both at the beginning and end of the month. During the month no experiments or extra clean-up was carried out. In making the test we allowed the old process to take out all it could take, *i.e.*, we ran the vats (by the old process) so long as sodium sulphide would show the least trace of silver in the solution.

When the old process would not take any more silver out, the extra solution of the Russel process was applied, and, as in the case of the old process, was run as long as sodium sulphide showed any trace of silver."

The actual clean-ups from the two processes were as follows:—

	Ounces.
Extracted by the old leaching process	26,361·42
Additional extracted by the Russel process	5,088·76
Total produce	31,450·18
Per cent. of total produce extracted by old process ...	83·8
Per cent. of total produce extracted by Russel process	16·2

The total additional cost of the Russel process, or, in other words, the extra expense of producing 5,088·76 ounces over the cost of the old process was as follows:—Chemicals, 632·50 dollars; fuel, 21·21 dollars; extra help, etc., 155·60 dollars; total, 809·31 dollars (£166 14s. 7½d.). At the then price of silver, the 5,088·76 ounces equalled 5,852·07 dollars. Deducting the above expenses 5,042·76 dollars (£1,050 11s. 6d.) is left as the net profit per month due to the extra treatment.

Another test was carried out during the month of November, 1890. In this run the total cost of chemicals per ton was 95 cents (3s. 11½d.). Of the total number of ounces extracted, the old process took out 80·29 per cent., and the Russel process the remaining 19·71 per cent. The additional ounces of silver extracted by the Russel process over the old process were 7,653·7, or (with silver at 1·025 dollars (4s. 3¼d.) per ounce) 7,845·04 dollars, making a net profit due to the use of the Russel system of about 7,000 dollars per month.

All the tailings which have been produced at the Yedras mill, by the old process before the introduction of the Russel process in 1887, have now been treated by the latter. The tailings are brought from the old dumps where they were thrown out in former years, and are charged direct to the leaching vats without any drying, roasting, or other treatment. Like the charges of ore, they are leached with water in order to remove the small percentage of soluble salts present; this washing requiring about four hours. A small percentage of ordinary hyposulphite solution is then applied, since the volume of the extra solution is only enough to saturate the charge, and as it would become diluted to some extent with the wash-water if it followed it, the small volume of ordinary solution is interposed.

As in treating ore, this extra solution amounts to 13 cubic feet per ton. It is followed by more of the hyposulphite solution to extract any silver which has been made soluble by the extra solution, but which has not passed out of the charge with it, remaining mechanically held in the pulp.

The total quantity of tailings from the old leaching process at Yedras, which have been re-treated by the Russel process, is between 30,000 to 40,000 tons. The following table shows the results:—

Year.	Ounces.	Extraction in Assay Office. Old Process. Per Cent.	Extraction in Assay Office. Russel Process. Per Cent.	Apparent Extraction in Mill. Per Cent.	Actual Extraction by Russel Process. Per Cent.
1888 ...	19·49	37·40	62·70	60·14	62·74
1889 ...	17·23	32·17	57·20	55·95	60·14
1890 ...	13·46	38·48	49·26	48·37	46·84

In the above table, apparent extraction in mill is obtained by comparing the value of the final tailings from the Russel process with the old tailings, as charged to the leaching-vats (taking also into account any soluble salts).

Actual extraction in mill is obtained by comparing the clean-up in silver with the silver actually charged to the vats. The chemicals used per ton of ore during these three years were as follows:—

Year.	Hypo-sulphite.	Blue-stone.	Caustic Soda.	Sulphur.	Total Chemicals per Ton.	Cost of Chemicals per Ton.	Ounces of Silver Extracted per Ton.
	Lbs.	Lbs.	Lbs.	Lbs.	Lbs.	Dollars. s. d.	
1888	2·08	6·32	5·54	3·30	16·24	1·45 6 0½	12·23
1889	1·62	5·11	3·13	2·69	12·55	1·12 4 8	9·75
1890	1·07	4·06	2·48	1·71	9·32	0·82 3 5	6·30

For a description of the Russel process at the Marsac mill, Park City, Utah, the writer is indebted to an article by Mr. W. G. Lamb in *The Engineering and Mining Journal*, New York, of December 17th, 1892.* It was started at these works on January 1st, 1889, superseding amalgamation at the end of that year. The statistics of amalgamation to which reference is made are from the Ontario mill in the same camp.

In that mill, amalgamation has been in continuous use since its start in January, 1887. As the wages and prices of fuel and supplies are the same for the two mills, a comparison of statistics is of value in determining the general efficiency and economy of the two processes, dealing, it should be added, with the same classes of ore, as the accompanying analyses indicate. The properties of the two companies, the Ontario and Daly, adjoin, and are in fact on the same vein. The equipment of the two mills and the staff employed are as follows :—Ontario, 2 rock-breakers, 2 rotatory driers, 40 ore-stamps, 10 salt-stamps, 2 Stetefeldt furnaces, 24 pans, 12 settlers, 71 mill men. Marsac, 1 rock-breaker, 2 rotatory driers, 30 ore-stamps, 5 salt-stamps, 1 Stetefeldt furnace, 6 (16½ feet) ore-vats, 8 (9 feet) precipitating-tanks, 51 mill men.

In the above connexion it is to be noted as before remarked that the labour item compared with figures given elsewhere appears unusually high at the Ontario for a 40 stamp amalgamating-battery, and correspondingly low for a lixiviation-plant of the same size, if the whole of the staff be included in both.

The analyses and values of the ore treated at the Ontario and Marsac mills for 1891 are as follows (the samples on which these analyses were made being composed of all the battery samples taken each day during the entire year) :—

	Ontario.	Marsac
Silica	75·0	76·60
Zinc	5·73	5·30
Lead	1·80	3·50
Iron	2·80	1·65
Sulphur	2·23	0·70
Lime	1·76	1·32
Magnesia	0·23	trace.
Copper	0·29	0·39
Silver (ounces)	39·50	39·10
Gold (ounces)	0·044 ($0·91)	0·044 ($0·91 = 3s. 9½d.)

* Page 580.

The following table gives the crushing statistics for 1891 :—

Mill.	Ore.	Time Battery run.	Mesh of Screen.	Rate of Crushing per Day.	Rate of Crushing per Stamp, per Day.
	Tons.	Days.		Tons.	Tons.
Ontario (40 stamps) ...	25,650	341·8	26	75·0	1·87
Marsac (30 stamps) ...	24,214	347·0	20	70·0	2·33

The above difference in rate of crushing per stamp is probably not due entirely to difference in mesh of screen, for Ontario ore may not crush so fast as Daly, even in the same battery and with the same mesh of screen. Here again we have a matter which may affect the relative results (supposing one process replacing the other) in regard to cost.

The Ontario product in bars of bullion averaged 425 fine in silver and 0·250 fine in gold; it contained also 57·5 per cent. copper. The Daly precipitates, including those from the wash-water, but not the lead carbonate product, averaged 313 fine in silver and 0·260 fine in gold; they contained 15·3 per cent. copper.

The cost of marketing the product was 3·47 cents per ounce for Ontario bullion and 3·45 cents per ounce for Marsac sulphides. The price obtained was 97·55 cents per ounce for silver in Ontario bullion and 97 cents for that in Marsac sulphides. In the Marsac sulphides shipped, 20·67 cents per ounce was received for the contained gold, against nothing for that contained in Ontario bullion.

The following is the consumption of water, chemicals, iron, and power per ton of ore at the Ontario and Marsac mills :—

Mill.	Water.	Chemicals and Mercury. Cost.	Iron.	Power.*	Machinery Expenses.
	Cubic Feet.	Dollars.	Lbs.	Horse-power.	Dollars.
Ontario	400	1·315	5·5	108	0·31
Marsac	56	0·924	0·05	8	0·07

The consumption of chemicals has increased since 1890 owing to the adoption of hot-solutions. The production increased 2·8 per cent. however, and while the total cost was increased 7,339·98 dollars the net gain in extraction after deducting the extra cost amounted to 18,478·92 dollars.

The following are some additional details of work at the Marsac and Ontario mills in 1891, when the 30 stamp Marsac mill crushed 24,215 tons of ore through a 20 mesh screen against 25,650 tons through a 26 mesh screen at the 40 stamp Ontario mill :—

Mill.	Fuel. Per Ton. Dollars.	Salt used in Roasting. Per Cent.	Labour.† Dollars.	Extraction of Silver. Per Cent.
Ontario (cords of wood)	0·153	13·9	0·46	90·8
Marsac (tons of coal)	0·087	8·26	0·31	91·8

* This is for power for driving pans and settlers at the Ontario, and for stirring and handling solution and grinding sulphides at Marsac.

† Includes that on the pans, amalgam, and bullion at Ontario; and on vats and shipment of sulphides at Marsac.

During 1892, up to December 1st, the percentage of salt used at Ontario had been increased to 14·2 per cent., and that at Marsac 9·5 per cent. The extraction at the Ontario mill remained at 90·8 per cent., and that at the Marsac 91·9 per cent. It is probable that the extraction at the latter works would be increased if the cooling floor-space were enlarged, so as to allow the ore to cool without wetting down. This would increase the expense, it is estimated, by only 13 cents per ton.

The following is the detailed annual cost of the lixiviation department at the Marsac mill :—

	Dollars.	Dollars.	s.	d.
Labour—				
1 foreman, $5·00; 3 leachers, $4·00; 3 shovellers, $3·00; 1 pressman, $3·50; 1 labourer, $1·50	16,790·00	... 0·6934 =	2	10¾
Chemicals—	Dollars. Dollars.			
Hyposulphite ... 152,808 lbs. at ·0362 = 5,531·64				
Bluestone ... 78,569 lbs. at ·0641 = 5,036·27				
Caustic ... 119,741 lbs. at ·0555 = 6,645·62				
Sulphur ... 80,486 lbs. at ·0257 = 2,068·49				
Soda ash ... 22,309 lbs. at ·0317 = 707·19				
	19,989·00	... 0·8255 =	3	5¼
Repairs—				
1 machinist, $4·00; materials and supplies, $2·50 per day	2,372·50	... 0·0979 =	0	4¾
Power	3,139·89	... 0·1297 =	0	6½
Assay office	2,008·82	... 0·0830 =	0	4¼
Total cost	$44,350·21	... $1·8295 =	7	7½

Figures for the following comparison of the Ontario and Marsac results for 1891 are taken from the respective reports of the Ontario and Daly mining companies :—

	Dollars.	Dollars.	£	s.	d.
Ontario—					
Cost of milling per ton	8·93				
Product expense	1·23				
		10·16 =	2	2	4
Marsac—					
Cost of milling per ton	6·27				
Product expense	1·23½				
		7·50½ =	1	11	3
Difference in favour of Marsac		2·66 =	0	11	1
Ontario—Mill extraction	91·00 %				
Marsac ,, ...	91·57 ,,				
Difference in favour of Marsac ...	0·57 ,,	= 0·21½ =	0	0	10¾
Ontario—Realized from gold	0·00				
Marsac ,, ,, ...	0·63½				
Difference in favour of Marsac ...		0·63½ =	0	0	7¾
Total difference claimed in favour of Russel process		3·51 =	0	14	7½

This total difference of 3·51 dollars would, it is asserted, have made a saving of 91,057 dollars, had the Ontario ore, amounting to 25,650 tons in 1891, been treated by the Russel process. Mr. Lamb sums up by saying that to treat about the same number of tons of ore per day, of approximately the same composition at the Marsac mill, the Ontario mill requires 39 per cent. more labour, 30 per cent. more stamps, more power, twice the number of furnaces, 48 per cent. more salt, and 40 per cent. greater cost of chemicals, and yields a smaller percentage of both gold and silver than the Marsac mill using the Russel process.

Whilst this may be true, a slight difference in the composition of the ore may affect the extraction, and looking at the respective analyses given, it seems hardly fair to assume that, even in the case of Ontario and Marsac ores, the commercial results of the lixiviation treatment will be exactly the same in both cases.

A comparison of extraction by lixiviation (using the Russel process) with that by amalgamation made in 1891 at the new works of the Blue Bird mine, Montana,* showed that the extraction by amalgamation varied from 58·5 per cent. to 80 per cent., while with lixiviation it averaged 84·1 per cent. It is claimed that the extraction might have been 8 per cent. higher (judging from experience elsewhere) if the lixiviation charges had not been wetted down hot.

The analysis of the battery samples for six months showed the ore to contain :—

	Per Cent.
Silica	64·4
Sulphur	5·0
Iron	3·74
Lead	4·22
Zinc	12·8
Manganese	5·21
Copper	0·20

The cost of chemicals and quicksilver averaged 0·80 dollars for amalgamation and 0·99 for lixiviation, using 3·3 to 3·7 lbs. of hypo-sulphite of soda, 5·6 to 9 lbs. of bluestone, 3·7 to 6·7 lbs. of soda ash, 4·4 to 5 lbs. of caustic soda, and 2·5 to 3·4 lbs. of sulphur, making a total of 18·8 to 27·5 lbs. of chemicals per ton of ore treated.

The leaching vats were filled with 20 to 70 tons charges about $7\frac{1}{2}$ feet deep. The first part of the wash-water was run in from below the filter, while the ore was being charged into the vat, the leaching with water being afterwards introduced from below downwards as soon as the ore

* C. A. Hoyt, *The Engineering and Mining Journal*, New York, Jan. 7th, 1893, page 8.

was charged. The base-metal leaching was followed by about 100 inches in depth of ordinary solution. This was succeeded by about 30 inches of extra solution, containing 1 per cent. of bluestone, which was allowed to stand seven to ten hours. This was again followed by 40 to 60 inches of ordinary solution, and then by 10 inches of extra solution of the same strength as before, which was allowed to stand seven to ten hours, and finally 50 to 60 inches of ordinary solution was run in and was expelled by the second wash-water.

The strength of the stock solution was 1·6 per cent. to 1·9 per cent. of hyposulphite of soda, whilst the extra solution contained in addition 1 per cent. of bluestone.

All solutions were kept at a temperature of 90 to 120 degs. Fahr., 11 to 15 per cent. of salt was used in roasting, and the ore was crushed on the average to pass a 24 mesh screen, the salt to 20 mesh.

The silver and gold were precipitated from both solutions and wash-water with sodium sulphide ; and the lead by itself from the solutions by soda-ash.

Mr. H. Lang, in a letter to *The Engineering and Mining Journal*,* New York, of March 18th, 1893, remarks :—"The Russel Company summarizing their claims declare that the Russel process, both metallurgically and economically, occupies the place formerly held by :—1st, the Kiss-patera, or old leaching process; 2nd, amalgamation of silver and silver-gold ores ; 3rd, smelting of dry ores, and ores averaging not over 15 per cent. lead, or such as do not contain sufficient lime or iron to make them desirable as fluxes in smelting."

Without expressing an opinion on the merits of the first two claims, Mr. Lang takes emphatic exception to the third and says :—"In no case and under no conditions can the Russel process treat basic ores as cheaply or as efficiently as can the smelting processes. With acid ores I recognize in full its advantages, but even with the most siliceous material it is questionable if the process can always compete with matting, even when silver alone is treated ; and when gold, copper, and other metals are worked for, it has no chance whatever."

In 1889, a company of Oregon capitalists erected a rather complete Russel process mill at Mineral, Idaho (a quarter of a mile below the spot where a matting plant now stands) at a cost of £6,250, which was under the charge of a skilful leaching-expert, Mr. W. H. Lamb.

Mr. Lamb laboured arduously and intelligently through several months ; but, in vain, the project was a failure, and the mill was shut down and sold for a tenth of its cost for other purposes.

* Page 244.

According to Mr. Lang's information, the best results reached 76 per cent., though it does not appear whether this was the extraction, apparent extraction, or simply an extra extraction. He goes on to add that the ores in question (of which only the more tractable part were sought to be worked in the Russel mill) have been bought by his firm during the last three years, and successfully treated in matting furnaces. That they have made it pay where the Russel process failed, is a sufficient answer to the assertion that the process occupies the place of smelting.

Treating the same ores, he claims to be able to do the work at one-fifth the cost, and save 20 per cent. more silver than the Russel process claimed to extract.

The high cost of treatment at these works is stated, however, to have arisen largely from faults of construction and design.

The claims of the process for the treatment of basic ores are only relatively true. The Russel Company quote an analysis of the Las Yedras ore (Sinaloa, Mexico) and say "smelting being economically and *metallurgically** out of the question." (This was said of an ore containing silica, 25 per cent.; calcite, 46 per cent.; iron, 9·8 per cent.; sulphur, 12·5 per cent.; and arsenic, 2·5 per cent.) Instead of being an unsmeltable combination, this ore is in reality the finest smelting product in the world, susceptible of being run down at one operation into a high-grade matte, and at less than the cost of the salt which is now used in roasting. And the matte can then be refined, and its total silver extracted at an additional cost per ton of original ore, not exceeding the cost of the chemicals now used in the Russel leaching. Mr. Lang bases this opinion on the data given in the valuable series of papers by Mr. Rockwell, published in *The Engineering and Mining Journal*,† New York, "On Roasting, Chloridizing, and Lixiviation at Yedras mine, Mexico." The Yedras ore is chemically nearly the same as that of an important mine near Mineral Hill, of which considerable quantities have been treated by Mr. Lang's firm, the difference being an excess of carbonate of lime in the Mexican ore. This is run down without the use of fluxes and without admixture of other ore, using 7 per cent. of coke.

Employing a furnace of special construction and by peculiar treatment of the blast, etc, the larger part of the sulphur and arsenic is burnt off, and the corresponding proportion of iron and zinc is slagged off, effecting a desirable concentration of the matte and at the same time utilizing the heat of combustion of the elements named. This is pyritic smelting, properly so-called, a branch of the larger art of matte smelting.

* The italics are mine. † Vol. xlv., pages 86, 106, 159, 178, 197, 213, and 283.

The waste of silver at Las Yedras must have been prodigious. Mr. Rockwell mentions months of work in which the losses by volatilization varied from 17 to 25 per cent., the best work attainable in his time resulting in an average loss of 10 per cent. from that cause. The Russel company have recorded it at 6 or 7 per cent. The total losses now amount to about 19 per cent.

The Russel process has undoubtedly been a great improvement over the old leaching with certain classes of ore, but it is a great wonder that the management tolerated any lixiviation methods whatever.

Mr. Lang states that he holds the same opinion concerning the Aspen* and Marsac works. The composition of the Aspen ore (30,000 tons of which have been treated by the Russel process) is lead, 2·27 per cent.: silica, 21·66 per cent.; sulphate of barium, 20·92 per cent.; lime, 10·99 per cent.; magnesia, 4·24 per cent.; iron, 10·02 per cent.; zinc, 2·85 per cent.; copper, 16 per cent.; sulphur, 8·10 per cent.; and arsenic, traces.

* The probable cost of pyritic smelting in this district would be under 12s. 6d. per ton, treating not less than 100 tons per day.

To illustrate Mr. Edward Halse's Paper on "The Gold-bearing Veins of the Organos District, U.S. Colombia".

Fig. 1.

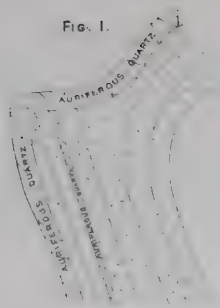


Fig. 2.

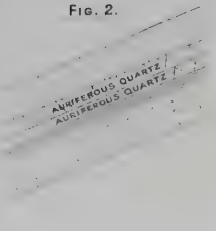


Fig. 3.

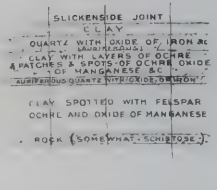

SLICKENSIDE JOINT
CLAY
QUARTZ WITH OXIDE OF IRON &c
CLAY WITH LAYERS OF OCHRE
& PATCHES & SPOTS OF OCHRE OXIDE
OF MANGANESE &C
AURIFEROUS QUARTZ WITH OXIDE OF IRON
CLAY SPOTTED WITH FELSPAR
OCHRE AND OXIDE OF MANGANESE
ROCK (SOMEWHAT SCHISTOSE)

Fig. 4.

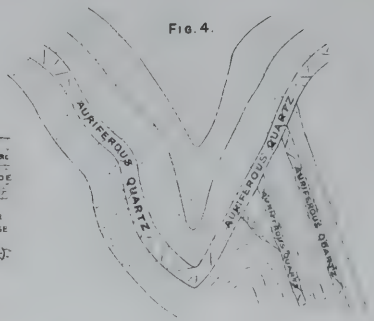


Fig. 5.

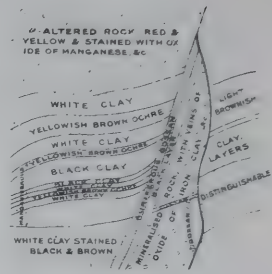


Fig. 6.

Fig. 7.

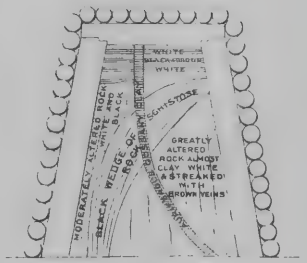

WHITE BLACK & BLUE WHITE
MODERATELY ALTERED ROCK WHITE & BLACK
BLACK WEDGE OF ROCK
SCHISTOSE
GREATLY ALTERED ROCK ALMOST CLAY WHITE & STREAKED WITH BROWN VEINS

Fig. 8.

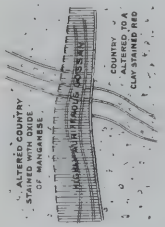


Fig. 9.

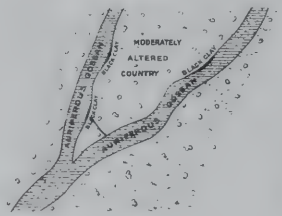


Fig. 10.

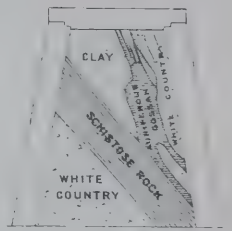


Fig. 11.

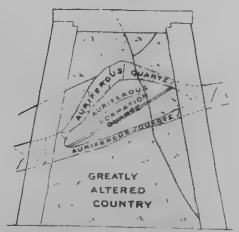


Fig. 12.

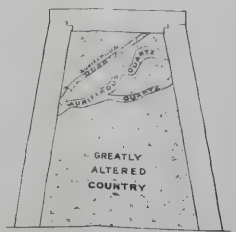
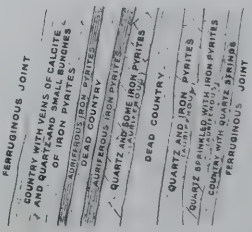


Fig. 13.

Fig. 14.


FERRUGINOUS JOINT
COUNTRY WITH VEINS OF CALCITE & QUARTZ AND SMALL BUNCHES OF IRON PYRITES
AURIFEROUS IRON PYRITES
DEAD COUNTRY
AURIFEROUS IRON PYRITES
QUARTZ AND SOME IRON PYRITES
DEAD COUNTRY
QUARTZ AND IRON PYRITES
QUARTZ SPRINKLED WITH IRON PYRITES
COUNTRY WITH QUARTZ STRINGS
FERRUGINOUS JOINT

Scale 4 Feet to 1 Inch

Wm. Brand & Son's Photo-Mech. Process.
Printed March 1st 1887

And.w Reid Colour Printer Newcastle

Vol. XLII Plate X

To illustrate Mr Edward Halse's Paper on "The Gold-bearing Veins of the Organos District U. S. Colombia".

Fig. 15.

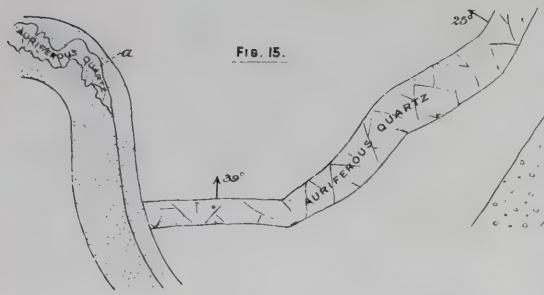


Fig. 16.

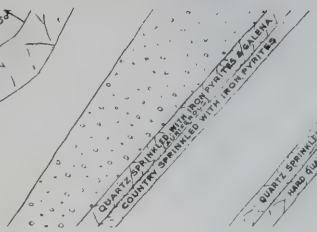


Fig. 17.

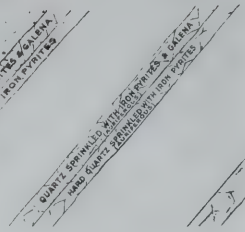


Fig. 18.

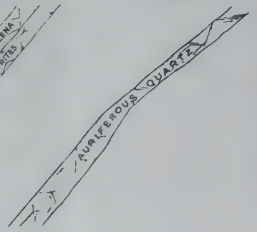


Fig. 19.

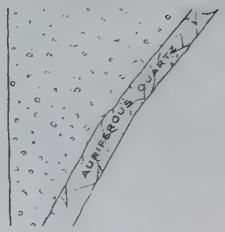


Fig. 20.

CLEARLY DEFINED JOINT WITH CLAY
REDDISH, BROWN & GREEN COUNTRY WITH QUARTZ AND IRON PYRITES
QUARTZ WITH PYRITES AND GALENA (AURIFEROUS)
GREEN COUNTRY WITH SOME VEINS OF QUARTZ AND IRON PYRITES

Fig. 21.

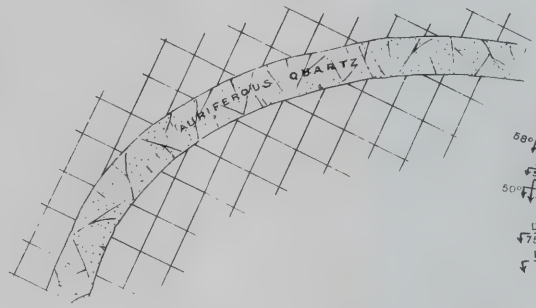


Fig. 25.

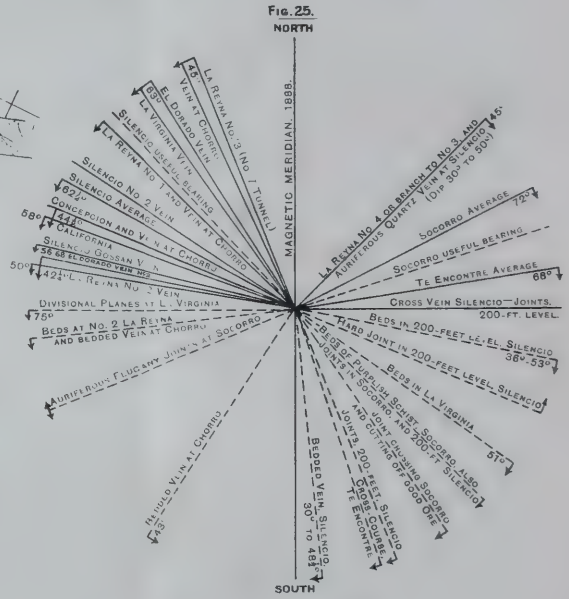

NORTH
MAGNETIC MERIDIAN, 1888.
45° LA REINA No. 3 AND / TUNNEL
VEIN AT CHORRO
65° EL DORADO VEIN
LA VIRGINIA VEIN
SILENCIO USEFUL BEARING
LA REINA No. 1 AND VEIN AT OTERO
SILENCIO No. 2 VEIN
62½° SILENCIO AVERAGE
CONCEPCION AND VEIN AT CHORRO
58° CALIFORNIA
SILENCIO GOSSAN VEIN
56° 48 m. EL DORADO VEIN No. 2
50° 42½° L. REINA No. 3 VEIN
DIVISIONAL PLANES AT L. VIRGINIA
75° BEDS AT No. 2 LA REINA AND BEDDED VEIN AT CHORRO
AURIFEROUS FLUCANY JOINT AT SOCORRO
LA REINA No. 4 OR BRANCH TO No. 3 AND AURIFEROUS QUARTZ VEIN AT SILENCIO (DIP 20° TO 30°)
45°
SOCORRO AVERAGE 72°
SOCORRO USEFUL BEARING
TE ENCONTRE AVERAGE 68°
CROSS VEIN SILENCIO—JOINTS.
200-FT. LEVEL
BEDS IN 200-FEET LEVEL SILENCIO 38°-53°
HARD JOINT IN 200-FEET LEVEL SILENCIO
DIP OF AURIFEROUS SCHIST, SOCORRO AT 200-FT. LEVEL
BEDS IN LA VIRGINIA
51°
JOINTS IN SOCORRO AND 200-FT. SILENCIO
JOINT CROSSING SOCORRO AND 200-FT. SILENCIO
VEIN AT CHORRO
45° BEDDED VEIN SILENCIO
30° TO 45°
CROSS COURSE
TE ENCONTRE
SOUTH

Fig. 22.

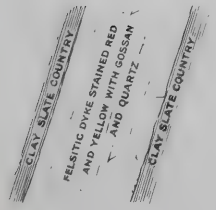


Fig. 23.

Fig. 24.

Scale 4 Feet to 1 Inch

REFERENCE.

Strike and Dip of Veins ——→

Strike and Dip of Joints, Beds or Bedded Veins - - - - →

And.w Reid Sons & Co. Printers, Newcastle

APPENDICES.

I.—BAROMETER, THERMOMETER, Etc., READINGS FOR THE YEAR 1892.

By M. WALTON BROWN.

The barometer, thermometer, etc., readings have been supplied by permission of the authorities of the Glasgow and Kew Observatories, and give some idea of the variations of temperature and of atmospheric pressure in the intervening districts in which the mining operations of this country are chiefly carried on.

The barometer at Kew is 34 feet, and at Glasgow is 180 feet, above sea-level. The barometer readings at Glasgow have been reduced to 32 feet above sea-level, by the addition of 0·150 inch to each reading, and the barometer readings at both observatories are reduced to 32 degs. Fahr.

The fatal explosions in collieries are obtained from the annual reports of H.M. Inspectors of Mines, and are printed upon the diagrams (Plates XVII., XVIII., XIX., and XX.) recording the meteorological observations.

JANUARY, 1892.

KEW.								GLASGOW.							
	Barometer.				Temperature.				Barometer.				Temperature.		
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon	Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon
1	29·736	29·844	29·940	30·083	43·2	35·7	W	1	29·593	29·783	29·948	30·032	41·2	35·9	WNW
2	30·184	30·236	30·157	30·104	41·8	31·8	WSW	2	29·958	29·910	29·799	29·736	44·0	38·4	W
3	30·027	30·030	30·015	30·019	43·5	31·9	NW	3	29·822	29·867	29·872	29·953	38·1	31·6	WNW
4	30·016	30·054	30·057	30·112	35·1	27·0	NW	4	30·002	30·074	30·050	29·946	36·1	29·0	NW
5	30·063	29·954	29·690	29·541	42·5	24·9	SW	5	29·617	29·430	29·379	29·311	43·6	35·0	WNW
6	29·490	29·357	29·399	29·450	44·2	32·8	NW	6	29·206	29·170	29 220	29·300	37·3	28·7	NW
7	29·432	29·401	29·323	29·273	37·0	30·5	SW	7	29·280	29·202	29·167	29·251	34·1	24·0	SSW
8	29·306	29·447	29·437	29·401	34·7	28·4	W	8	29·251	29·233	29·323	29·459	34·3	29·7	W
9	29·364	29·435	29·428	29·423	32·7	24·9	NNW	9	29·496	29·537	29·502	29·532	33·9	25·9	N
10	29·436	29·533	29·581	29·673	32·1	24·1	N	10	29·558	29·746	29·889	30·000	36·0	29·4	NE
11	29·761	29·817	29·851	29·912	35·8	29·8	N	11	30·034	30·013	29·917	29·845	36·0	27·2	SW
12	29·895	29·916	29·823	29·814	35·4	24·3	N	12	29·770	29·825	29·867	29·871	37·8	29·6	E
13	29·781	29 750	29·630	29·573	36·8	30·7	NE	13	29·789	29·643	29·446	29·370	35·3	27·9	NE
14	29·491	29·480	29·439	29·460	34·5	28·9	N	14	29·300	29·298	29·293	29·325	36·3	33·2	S
15	29·464	29·491	29·475	29·490	34·3	25·2	NW	15	29·386	29·411	29·396	29·387	33·9	26·3	E
16	29·457	29·488	29·473	29·494	36·3	23·8	SE	16	29 350	29·368	29·379	29·471	32·1	24·8	E
17	29·493	29·565	29·573	29·608	37·5	32·7	E	17	29·550	29·593	29·608	29·636	37·8	30·8	ESE
18	29·640	29·734	29·770	29·833	44·0	36·8	E	18	29·651	29·687	29·702	29·727	37·8	35·5	ENE
19	29·831	29·847	29·827	29·843	41·2	32·8	E	19	29·745	29·825	29·842	29·817	38·0	33·9	NE
20	29·833	29·850	29·863	29·931	37·7	32·4	E	20	29·780	29·810	29·861	29·894	35·2	33·9	E
21	29·933	29·947	29·892	29·877	36·4	30·4	E	21	29·839	29·720	29·512	29·478	40·0	34·1	SSW
22	29·866	29·845	29·827	29·915	45·7	34·9	S	22	29·575	29·610	29·616	29·679	39·9	34·6	SSW
23	29·909	29·963	29·932	29·972	49·3	41·4	SW	23	29·760	29·761	29·615	29·623	45·3	38 9	S
24	29·977	30·092	30·160	30·256	49·3	33·9	NW	24	29·726	29·883	29·985	30·061	48·5	37·2	W
25	30·317	30·418	30·448	30·477	42·3	31·8	NW	25	30·155	30·280	30·316	30·322	43·2	35·8	WSW
26	30·493	30·474	30·358	30·274	43·5	29·8	SW	26	30·205	30·119	30·064	29·951	46·8	40·2	SW
27	30·180	30·092	29·847	29·878	47·2	41·6	SW	27	29·770	29·547	29·537	29·746	47·8	35·9	SW
28	30·111	30·277	30·244	30·171	47·8	36·7	WSW	28	29·883	29·820	29·741	29·659	48·5	35·8	SW
29	30·057	30·085	30·149	30·197	51·9	47·8	W	29	29·635	29·672	29·799	29·765	51·2	48·4	SW
30	30·175	30·179	30·106	30·096	50·8	46·8	W	30	29·768	29·800	29·778	29·820	48·4	40·3	SW
31	30·050	30·134	30·103	29·987	49·8	45·6	N	31	29·847	29·919	29·816	29·470	46·6	39·4	SW

FEBRUARY, 1892.

1	29·766	29·537	29·416	29·382	48·0	37·5	SW	1	29·107	28·862	28·801	28·846	46·6	34·9	SW
2	29·226	29·328	29·223	29·153	42·4	33·7	W	2	28·858	28·789	28·617	28·914	40·7	29·9	WSW
3	29·143	29·264	29·402	29·614	43·5	35·8	WNW	3	29·082	29·272	29·426	29·556	40·0	32·7	WNW
4	29·722	29·733	29·587	29·525	45·3	31·9	SSW	4	29·537	29·356	29·155	29·246	41·6	31·6	SSW
5	29·546	29·608	29 708	29·814	48·3	38·5	W	5	29·199	29·323	29·418	29·477	41·1	34·7	W
6	29·829	29·860	29·828	29·869	45·5	38·7	W	6	29·521	29·603	29·635	29·719	43·4	37·7	WSW
7	29·901	29·924	29·828	29·774	52·9	43·8	SW	7	29·712	29·568	29·393	29·528	48·4	40·0	W
8	29·797	29·683	29·778	30·037	50·5	44·5	NW	8	29·522	29·691	29·896	30·043	48·2	40·3	WNW
9	30·214	30·312	30·360	30·300	47·7	41·4	SSW	9	30·116	30·138	30·139	30·175	48·3	40·2	SW
10	30·278	30·324	30·325	30·383	49·2	44·0	NW	10	30·185	30·221	30·222	30·281	47·9	43·6	WSW
11	30·401	30·439	30·433	30·460	50·0	38·1	NW	11	30·295	30·342	30·368	30·390	49·3	44·3	W
12	30·457	30·471	30·434	30·452	43 4	32·9	NNW	12	30·374	30·420	30·409	30·469	48·7	41·8	WSW
13	30·453	30·503	30·472	30·462	42·3	34·2	N	13	30·524	30·564	30·491	30·400	44·1	38·8	NE
14	30·366	30·214	29·918	29·717	41·0	32·3	SW	14	30·193	29·945	29·682	29·581	44·4	37·4	SW
15	29·490	29·431	29·445	29·579	41·9	29·7	WNW	15	29·616	29·694	29·761	29·822	38·3	31·4	ENE
16	29·595	29·634	29·602	29·576	30·8	23·7	NE	16	29·857	29·898	29·803	29·678	34·9	28·2	NE
17	29·528	29·490	29·336	29·106	33·5	19·2	W	17	29·405	29·243	29·183	29·167	36·1	30·3	E
18	29·089	29·176	29·267	29·350	32·7	23·0	NW	18	29·243	29·275	29·280	29·368	33·1	22·4	NW
19	29·337	29·319	29·206	29·182	34·0	23·5	E	19	29·394	29·412	29·409	29·460	30·4	14·8	ESE
20	29·191	29·351	29·423	29·423	37·0	28·8	NE	20	29·473	29·474	29·497	29·554	33·5	24·0	NE
21	29·341	29·411	29·438	29·461	43·8	36·1	S	21	29·491	29·419	29·365	29·432	36·2	31·1	ENE
22	29·452	29·543	29·578	29·642	50·5	38·2	S	22	29·431	29·446	29·445	29·520	39·8	33·3	NE
23	29·651	29·646	29·656	29·735	48·8	36·1	SE	23	29·562	29·620	29·643	29·678	45·0	35·2	E
24	29·756	29·778	29·780	29·848	48·0	34·7	NE	24	29·681	29·730	29·718	29·757	42·7	35·2	ENE
25	29·854	29·883	29·880	29·941	52·2	37 1	E	25	29·774	29·820	29·827	29·882	40·2	36·0	E
26	29·953	30·049	30·067	30·106	42·1	36·2	ENE	26	29·927	30·009	30·025	30·094	42·9	35·5	ENE
27	30·086	30·108	30·038	29·994	40·9	34·3	NE	27	30·117	30·130	30·101	30·090	39·2	35·9	ENE
28	29·939	29·890	29·830	29·833	41·9	41·4	NE	28	30·038	30·019	29·949	29·962	44·1	34·3	E
29	29·804	29·831	29·831	29·863	34·9	35·3	N	29	29·966	30·000	30·019	30·071	40·3	34·5	NE

MARCH, 1892.

KEW.								GLASGOW.							
	Barometer.				Temperature.		Direction of wind at noon		Barometer.				Temperature.		Direction of wind at noon
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.		Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	
1	29·826	29·883	29·888	29·896	39·9	33·5	NNE	1	30·080	30·127	30·136	30·212	40·0	34·7	NE
2	29·954	30·050	30·091	30·148	34·0	28·2	NE	2	30·240	30·294	30·304	30·344	35·2	32·9	ENE
3	30·177	30·234	30·219	30·243	31·8	28·1	NE	3	30·348	30·365	30·342	30·360	34·1	31·8	ESE
4	30·241	30·284	30·298	30·332	32·9	29·1	NE	4	30·357	30·387	30·342	30·356	38·2	29·4	N
5	30·314	30·303	30·238	30·230	39·4	26·7	N	5	30·339	30·325	30·257	30·250	36·1	28·5	E
6	30·199	30·201	30·085	30·036	35·8	31·1	E	6	30·199	30·168	30·103	30·078	38·8	31·2	NE
7	29·964	29·958	29·896	29·901	36·6	29·3	E	7	30·005	29·977	29·906	29·901	38·2	31·8	S
8	29·888	29·906	29·855	29·865	34·5	25·8	NNE	8	29·845	29·819	29·699	29·588	41·5	31·1	WSW
9	29·754	29·557	29·462	29·476	38·9	22·2	WSW	9	29·439	29·407	29·355	29·166	33·1	25·4	WNW
10	29·423	29·313	29·278	29·305	36·4	24·6	WSW	10	29·193	29·401	29·528	29·648	38·0	27·5	NW
11	29·502	29·565	29·580	29·607	38·1	26·9	N	11	29·650	29·650	29·656	29·686	39·5	30·0	ENE
12	29·572	29·533	29·434	29·326	37·7	27·2	NNW	12	29·648	29·591	29·467	29·463	38·2	29·5	N
13	29·350	29·348	29·327	29·308	40·5	24·9	W	13	29·355	29·353	29·312	29·338	38·9	30·5	NE
14	29·377	29·464	29·540	29·656	39·1	25·8	W	14	29·320	29·361	29·388	29·429	39·2	28·9	WSW
15	29·674	29·599	29·273	29·269	43·8	31·5	S	15	29·406	29·320	29·260	29·423	38·8	27·5	SE
16	29·619	29·898	29·968	30·032	51·5	37·3	W	16	29·611	29·708	29·671	29·654	44·4	26·7	S
17	30·063	30·141	30·186	30·265	56·0	45·3	SW	17	29·631	29·669	29·800	29·918	50·3	43·9	SSW
18	30·290	30·316	30·241	30·244	59·2	40·8	SE	18	29·998	30·101	30·114	30·138	57·7	47·2	SSE
19	30·218	30·208	30·144	30·150	52·3	38·0	ENE	19	30·154	30·210	30·186	30·255	58·8	36·9	SE
20	30·122	30·123	30·141	30·233	48·2	32·1	E	20	30·238	30·232	30·153	30·214	46·4	34·5	NE
21	30·273	30·343	30·284	30·332	51·6	27·5	E	21	30·232	30·293	30·297	30·405	46·2	31·2	W
22	30·328	30·387	30·407	30·483	41·7	37·7	N	22	30·486	30·573	30·558	30·570	48·1	35·2	N
23	30·482	30·478	30·370	30·364	47·2	34·1	NE	23	30·501	30·464	30·373	30·343	53·3	39·2	WNW
24	30·310	30·295	30·218	30·172	44·2	36·7	NE	24	30·253	30·204	30·131	30·071	48·9	37·7	W
25	30·099	30·036	29·864	29·792	49·8	36·0	SE	25	29·959	29·883	29·745	29·633	44·7	40·0	WSW
26	29·685	29·669	29·613	29·635	57·0	37·6	W	26	29·543	29·531	29·610	29·743	43·6	28·0	SW
27	29·643	29·710	29·720	29·781	49·6	38·9	W	27	29·823	29·911	29·936	29·988	35·1	26·2	NE
28	29·820	29·908	29·996	30·140	39·0	32·0	N	28	29·962	29·958	30·044	30·266	40·0	24·2	WSW
29	30·256	30·416	30·457	30·539	40·0	31·2	N	29	30·377	30·464	30·472	30·498	41·2	23·3	SW
30	30·528	30·551	30·488	30·543	49·3	32·6	NE	30	30·504	30·523	30·491	30·494	47·8	32·8	SSW
31	30·518	30·550	30·495	30·482	53·8	32·9	NE	31	30·456	30·446	30·399	30·399	55·1	35·6	SSW

APRIL, 1892.

1	30·466	30·455	30·362	30·391	65·0	30·0	N	1	30·394	30·415	30·377	30·385	60·8	39·1	W
2	30·386	30·370	30·302	30·310	62·3	35·9	E	2	30·360	30·355	30·293	30·291	63·7	33·7	NE
3	30·258	30·237	30·102	30·084	64·1	34·5	ENE	3	30·249	30·209	30·074	30·023	61·1	37·5	SSE
4	29·966	29·987	29·933	29·983	68·5	37·0	W	4	29·904	29·840	29·796	29·839	59·5	39·8	S
5	29·965	29·974	29·896	29·926	68·7	38·0	SSW	5	29·835	29·879	29·889	29·932	56·7	44·1	SSE
6	29·921	29·926	29·873	29·911	65·0	43·2	E	6	29·935	29·977	29·967	30·002	52·7	44·5	NE
7	29·893	29·907	29·859	29·927	64·3	42·6	NE	7	29·992	30·017	30·000	30·053	49·0	41·0	ENE
8	29·916	29·946	29·891	29·939	63·9	39·8	NE	8	30·060	30·088	30·026	30·095	53·8	40·0	E
9	29·923	29·934	29·895	29·925	59·8	41·9	NE	9	30·098	30·108	30·032	30·062	51·7	39·0	ENE
10	29·918	29·934	29·907	29·942	63·5	39·9	ENE	10	30·048	30·048	29·965	29·996	57·2	37·0	ENE
11	29·921	29·914	29·845	29·901	63·2	34·9	NNE	11	29·971	29·966	29·935	30·022	59·0	29·4	NE
12	29·894	29·895	29·822	29·771	48·2	33·9	NNE	12	29·994	29·983	29·883	29·844	43·0	35·4	E
13	29·656	29·642	29·606	29·643	39·7	34·0	NE	13	29·807	29·764	29·685	29·708	42·2	29·4	N
14	29·644	29·682	29·671	29·714	45·3	30·0	N	14	29·680	29·679	29·647	29·694	42·3	27·2	SE
15	29·704	29·743	29·667	29·610	48·0	27·9	W	15	29·688	29·703	29·674	29·679	39·5	25·9	SW
16	29·471	29·490	29·542	29·625	45·6	33·2	NNW	16	29·629	29·629	29·631	29·700	40·5	26·3	NE
17	29·648	29·700	29·728	29·864	46·8	29·8	N	17	29·717	29·745	29·747	29·853	46·7	25·4	W
18	29·926	30·023	30·048	30·179	48·1	31·3	NW	18	29·893	29·966	30·024	30·173	47·9	29·3	NW
19	30·271	30·360	30·356	30·402	50·8	31·3	WNW	19	30·221	30·240	30·174	30·098	49·8	28·7	SW
20	30·386	30·350	30·246	30·143	48·8	38·3	SW	20	30·065	30·078	29·926	29·822	56·1	42·5	SW
21	30·059	30·117	30·146	30·213	62·1	47·9	NW	21	29·863	29·922	29·874	29·845	51·9	43·9	SW
22	30·219	30·253	30·256	30·347	65·8	48·2	W	22	29·886	30·008	30·094	30·158	53·9	42·7	WSW
23	30·380	30·414	30·335	30·327	63·5	40·8	W	23	30·173	30·198	30·175	30·159	53·1	42·9	WSW
24	30·277	30·328	30·238	30·133	60·0	48·7	NNW	24	30·174	30·177	30·077	29·931	52·1	40·4	W
25	29·959	29·842	29·819	29·884	55·0	41·5	NW	25	29·808	29·788	29·761	29·784	51·0	37·4	NNW
26	29·863	29·848	29·814	29·863	54·3	36·8	NW	26	29·795	29·821	29·789	29·772	52·1	34·4	NNW
27	29·835	29·785	29·613	29·578	55·0	33·1	SW	27	29·600	29·388	29·357	29·580	48·3	36·3	NW
28	29·534	29·601	29·778	29·913	44·7	35·7	N	28	29·806	29·881	29·858	29·870	48·3	37·3	N
29	29·889	29·905	29·953	30·090	48·0	35·2	NNE	29	29·901	29·975	29·977	30·013	50·9	35·4	NNE
30	30·134	30·179	30·146	30·172	56·0	31·9	N	30	30·002	30·024	30·034	30·011	52·3	38·8	WSW

MAY, 1892.

KEW.								GLASGOW.							
	Barometer.				Temperature.				Barometer.				Temperature.		
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon	Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon
1	30·122	30·040	29·897	29·856	57·4	33·9	S	1	29·933	29·879	29·798	29·824	54·8	38·8	SW
2	29·805	29·800	29·772	29·779	48·1	39·9	NE	2	29·872	29·924	29·931	30·038	59·9	34·8	ENE
3	29·717	29·672	29·605	29·631	45·7	42·3	N	3	30·022	29·968	29·909	29·978	54·2	39·9	NE
4	29·669	29·735	29·801	29·916	48·5	41·5	NE	4	29·976	29·994	29·989	30·034	54·9	36·8	ESE
5	29·920	29·943	29·938	30·021	47·7	38·5	N	5	29·999	29·998	29·993	30·125	57·7	38·0	E
6	30·064	30·126	30·160	30·243	50·1	35·3	N	6	30·161	30·191	30·149	30·121	55·9	32·1	W
7	30·240	30·292	30·117	30·133	60·7	31·2	SW	7	30·030	29·972	29·956	29·978	53·6	39·7	WSW
8	30·139	30·150	30·106	30·135	63·4	39·9	W	8	29·993	30·018	30·018	30·047	56·1	41·8	W
9	30·152	30·160	30·121	30·170	65·8	37·2	SE	9	30·063	30·092	30·094	30·121	62·9	43·7	SW
10	30·169	30·161	30·124	30·181	64·0	41·0	ENE	10	30·155	30·174	30·162	30·278	67·3	39·2	NE
11	30·193	30·225	30·221	30·332	67·4	41·7	NE	11	30·343	30·366	30·335	30·391	60·3	40·4	ENE
12	30·359	30·355	30·285	30·304	68·0	42·1	NNE	12	30·407	30·366	30·247	30·218	64·2	39·6	NE
13	30·257	30·232	30·134	30·107	69·2	41·7	NNW	13	30·110	30·050	29·973	29·897	56·2	50·9	WSW
14	30·041	30·023	29·977	30·036	64·3	46·0	W	14	29·752	29·831	29·831	29·865	57·9	46·8	W
15	30·010	29·978	29·862	29·829	57·5	43·2	SW	15	29·636	29·552	29·548	29·522	57·4	46·4	W
16	29·792	29·756	29·691	29·674	61·9	45·0	W	16	29·382	29·283	29·298	29·546	53·1	44·7	WSW
17	29·749	29·883	29·960	30·071	61·5	48·4	NW	17	29·679	29·857	29·877	29·935	58·4	42·7	N
18	30·090	30·087	30·018	29·998	61·5	45·8	SW	18	29·899	29·748	29·701	29·717	55·2	41·2	WNW
19	30·000	30·009	30·067	30·070	62·4	46·9	W	19	29·702	29·791	29·810	29·650	56·9	42·5	W
20	29·998	29·845	29·843	29·927	61·8	48·9	W	20	29·544	29·561	29·724	29·914	57·0	39·3	WNW
21	30·023	30·063	29·970	29·979	64·3	44·0	W	21	29·901	29·828	29·783	29·814	53·7	38·2	WSW
22	29·961	29·983	29·937	29·936	66·2	43·4	SSW	22	29·870	29·901	29·837	29·767	51·2	39·5	SE
23	29·910	29·893	29·886	29·916	69·7	46·7	SW	23	29·643	29·655	29·637	29·602	57·2	43·0	W
24	29·905	29·896	29·862	29·860	71·4	51·9	S	24	29·534	29·577	29·641	29·699	62·1	50·7	SSW
25	29·740	29·716	29·752	29·805	75·2	53·2	S	25	29·689	29·580	29·550	29·666	60·9	48·5	ESE
26	29·781	29·795	29·766	29·776	75·3	55·1	ENE	26	29·715	29·782	29·739	29·705	64·8	48·4	W
27	29·766	29·836	29·856	29·846	71·7	59·0	S	27	29·651	29·705	29·752	29·837	66·0	53·3	WSW
28	29·729	29·690	29·696	29·884	77·9	52·6	S	28	29·837	29·809	29·677	29·543	61·0	49·3	ENE
29	29·928	29·954	29·988	30·008	66·9	51·1	SSW	29	29·443	29·616	29·617	29·673	62·8	48·9	SSW
30	30·071	30·108	30·071	30·030	73·5	55·2	SW	30	29·714	29·763	29·778	29·832	65·7	49·2	SSW
31	.....		29·848	29·714	29·759	80·7	55·4	S	31	29·774	29·711	29·649	29·563	68·5	55·0	SW

JUNE, 1892.

1	29·792	29·841	29·847	29·864	67·3	52·5	SW	1	29·470	29·508	29·512	29·623	62·2	47·5	SW
2	29·744	29·688	29·698	29·759	65·9	54·0	SW	2	29·558	29·260	29·291	29·437	54·2	45·9	S
3	29·826	29·915	29·949	30·021	65·9	50·3	SW	3	29·463	29·559	29·636	29·723	53·3	45·4	WSW
4	30·056	30·060	29·987	29·907	63·9	46·3	SW	4	29·752	29·778	29·713	29·583	58·9	45·9	SW
5	29·728	29·727	29·789	29·969	65·2	51·9	W	5	29·462	29·598	29·741	29·887	60·5	48·9	W
6	30·077	30·160	30·194	30·286	71·2	49·1	N	6	30·015	30·111	30·167	30·253	65·0	47·3	W
7	30·316	30·356	30·324	30·357	67·8	49·3	E	7	30·273	30·303	30·299	30·315	68·1	55·2	NW
8	30·356	30·379	30·321	30·314	69·3	48·0	E	8	30·308	30·311	30·274	30·275	76·1	53·0	WSW
9	30·273	30·231	30·157	30·106	74·8	53·2	E	9	30·243	30·214	30·116	30·058	77·3	53·8	W
10	30·020	29·923	29·816	29·792	80·7	51·1	S	10	30·015	29·995	29·918	29·857	60·2	45·8	E
11	29·773	29·778	29·747	29·779	68·3	55·0	SW	11	29·820	29·816	29·862	29·890	51·0	43·3	ESE
12	29·843	29·887	29·877	29·909	55·2	45·7	ENE	12	29·863	29·826	29·880	30·015	55·0	42·7	WNW
13	29·978	30·058	30·087	30·146	57·9	41·8	NNE	13	30·079	30·127	30·115	30·162	56·2	39·0	N
14	30·125	30·092	30·043	30·062	54·2	40·9	NE	14	30·191	30·185	30·086	30·076	62·2	39·0	SE
15	30·032	30·010	29·960	29·958	61·6	38·2	N	15	30·020	29·965	29·906	29·841	61·1	41·9	W
16	29·926	29·902	29·867	29·905	61·1	46·2	W	16	29·783	29·836	29·803	29·852	58·2	42·9	NNE
17	29·931	29·950	29·933	29·944	60·0	45·1	NW	17	29·847	29·824	29·881	29·949	57·4	39·4	NW
18	29·946	29·964	29·919	29·900	59·7	42·8	WNW	18	29·902	29·884	29·828	29·838	58·3	45·9	NE
19	29·870	29·838	29·799	29·827	63·3	45·2	SW	19	29·806	29·782	29·720	29·682	59·1	43·0	W
20	29·818	29·810	29·792	29·834	61·6	45·0	SW	20	29·641	29·703	29·726	29·795	59·0	47·3	E
21	29·853	29·886	29·903	29·933	63·9	46·0	SW	21	29·796	29·831	29·792	29·811	65·9	42·0	NNE
22	29·890	29·881	29·789	29·759	71·5	46·0	WSW	22	29·780	29·775	29·724	29·706	61·2	59·0	NW
23	29·583	29·500	29·641	29·815	62·9	51·0	NW	23	29·636	29·628	29·624	29·732	64·1	50·0	NW
24	29·911	29·965	29·948	29·976	70·5	46·5	S	24	29·769	29·824	29·834	29·861	65·0	46·9	W
25	29·961	29·954	29·897	29·880	66·3	51·6	S	25	29·820	29·747	29·628	29·610	58·0	48·4	WSW
26	29·958	30·020	30·032	30·073	75·0	54·6	WSW	26	29·684	29·715	29·671	29·684	67·0	54·0	SW
27	30·080	30·110	30·086	30·113	76·9	59·8	S	27	29·684	29·771	29·848	29·932	64·0	50·9	WSW
28	30·130	30·121	30·026	29·912	77·3	56·8	N	28	30·037	30·114	30·107	30·069	63·3	47·9	W
29	29·796	29·894	30·102	30·256	63·2	49·0	WNW	29	30·025	30·049	30·070	30·139	63·3	47·7	E
30	30·297	30·307	30·247	30·253	69·0	44·0	S	30	30·138	30·112	30·071	30·053	62·2	46·3	SW

JULY, 1892.

KEW.

Date.	Barometer. 4 A.M.	10 A.M.	4 P.M.	10 P.M.	Temperature. Max	Min.	Direction of wind at noon
1	30·243	30·237	30·181	30·188	71·1	48·2	SW
2	30·145	30·110	30·009	29·939	74·9	52·4	S
3	29·821	29·709	29·680	29·780	77·0	55·7	S
4	29·825	29·891	29·918	29·990	74·0	59·2	SW
5	30·001	29·989	29·908	29·858	66·9	52·3	SW
6	29·864	29·926	29·859	29·703	66·5	53·2	WSW
7	29·619	29·616	29·687	29·791	69·1	55·2	WSW
8	29·870	29·963	29·995	30·074	68·4	51·2	W
9	30·091	30·079	30·032	29·960	68·9	48·9	SSW
10	29·938	30·002	30·008	30·020	66·0	52·3	W
11	30·000	29·966	29·874	29·780	66·2	53·5	E
12	29·695	29·590	29·558	29·561	65·3	54·6	ENE
13	29·552	29·556	29·593	29·665	60·3	52·9	N
14	29·693	29·803	29·897	29·973	58·0	53·0	N
15	30·004	30·023	29·974	29·948	66·8	51·2	S
16	29·862	29·802	29·724	29·707	69·2	51·8	SE
17	29·686	29·747	29·799	29·884	59·8	51·3	NE
18	29·912	29·973	29·982	29·993	60·4	50·3	N
19	29·963	29·896	29·647	29·535	58·8	49·0	WSW
20	29·591	29·843	30·039	30·167	62·4	52·2	N
21	30·189	30·215	30·172	30·194	65·4	46·7	N
22	30·202	30·211	30·180	30·217	72·8	48·9	NW
23	30·223	30·232	30·210	30·250	73·1	53·4	NW
24	30·273	30·306	30·279	30·287	65·6	53·3	NE
25	30·263	30·261	30·182	30·180	62·1	53·6	NE
26	30·153	30·149	30·106	30·169	68·9	54·7	NE
27	30·169	30·183	30·176	30·212	66·4	54·1	NE
28	30·206	30·205	30·152	30·187	69·9	52·1	NNE
29	30·190	30·192	30·120	30·153	71·3	50·0	N
30	30·128	30·129	30·088	30·099	70·1	52·6	NNE
31	30·076	30·042	29·971	29·984	73·9	53·6	W

GLASGOW.

Date.	Barometer. 4 A.M.	10 A.M.	4 P.M.	10 P.M.	Temperature. Max	Min.	Direction of wind at noon
1	30·044	30·055	30·057	30·061	61·1	48·4	W
2	30·001	29·902	29·806	29·729	60·2	50·4	S
3	29·664	29·629	29·590	29·587	59·8	54·0	NW
4	29·596	29·709	29·784	29·741	60·9	50·0	W
5	29·546	29·542	29·563	29·617	60·1	50·1	WSW
6	29·616	29·584	29·280	29·180	62·1	48·5	SSW
7	29·106	29·220	29·378	29·381	59·0	48·9	W
8	29·427	29·543	29·657	29·744	60·0	49·0	WSW
9	29·796	29·850	29·821	29·837	59·1	50·0	SW
10	29·904	29·978	29·982	29·994	68·1	46·0	SE
11	29·962	29·928	29·854	29·852	66·0	49·8	E
12	29·808	29·766	29·731	29·710	56·0	50·8	E
13	29·697	29·699	29·725	29·827	61·7	49·1	E
14	29·850	29·894	29·881	29·909	59·0	49·9	ENE
15	29·933	29·953	29·937	29·945	58·0	50·0	NE
16	29·911	29·909	29·868	29·910	61·1	48·0	E
17	29·880	29·892	29·873	29·883	61·2	49·0	NNW
18	29·853	29·862	29·869	29·909	62·0	47·8	WSW
19	29·755	29·429	29·553	29·861	59·2	50·0	WNW
20	29·991	30·103	30·113	30·148	65·5	49·1	NW
21	30·153	30·151	30·122	30·144	64·8	44·9	W
22	30·150	30·186	30·194	30·210	61·1	51·2	W
23	30·222	30·242	30·240	30·284	70·0	51·0	W
24	30·296	30·305	30·273	30·282	71·0	49·8	SW
25	30·263	30·252	30·201	30·233	71·2	49·0	W
26	30·243	30·256	30·236	30·288	67·7	51·7	NE
27	30·296	30·296	30·286	30·311	60·8	50·9	E
28	30·294	30·290	30·233	30·262	68·1	51·3	NE
29	30·253	30·264	30·210	30·221	70·0	49·9	E
30	30·202	30·173	30·102	30·039	68·5	51·0	W
31	29·962	29·939	29·928	29·894	66·7	57·0	WNW

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1	29·934	29·894	29·932	30·038	68·6	56·2	NW
2	30·100	30·118	30·106	30·124	61·7	51·9	N
3	30·101	30·080	30·017	30·017	68·3	54·1	NW
4	30·011	30·040	30·032	30·098	66·2	51·2	NW
5	30·099	30·094	30·000	29·991	70·9	45·1	SW
6	29·941	29·948	29·920	29·989	70·2	52·0	SW
7	29·948	30·000	29·990	29·970	68·7	58·3	W
8	29·902	29·858	29·759	29·757	74·3	57·5	S
9	29·753	29·896	30·044	30·127	60·9	52·9	N
10	30·158	30·222	30·194	30·225	60·9	48·9	N
11	30·212	30·209	30·115	30·133	69·3	43·8	SW
12	30·123	30·100	29·977	29·962	71·2	49·0	SW
13	29·844	29·744	29·725	29·738	69·9	48·0	SW
14	29·798	29·832	29·816	29·706	70·8	57·4	SSW
15	29·771	29·911	29·939	30·054	72·5	57·7	SW
16	30·075	30·101	29·972	29·905	69·6	49·8	ESE
17	29·877	29·898	29·905	29·907	79·3	56·0	SSW
18	29·864	29·754	29·779	29·646	72·1	60·4	SE
19	29·675	29·691	29·724	29·809	65·2	60·0	N
20	29·900	30·024	30·068	30·139	71·0	52·2	WNW
21	30·164	30·203	30·145	30·147	72·6	49·1	S
22	30·076	30·009	29·902	29·869	75·3	46·8	SE
23	29·805	29·786	29·736	29·750	78·6	51·7	ESE
24	29·730	29·753	29·699	29·691	72·2	61·1	W
25	29·642	29·702	29·762	29·887	68·3	56·0	SW
26	29·926	30·003	29·982	29·999	67·8	55·0	SW
27	29·910	29·825	29·655	29·604	64·3	53·4	SSW
28	29·565	29·519	29·626	29·740	67·6	52·3	NW
29	29·737	29·686	29·613	29·600	70·1	52·2	SSW
30	29·577	29·596	29·527	29·552	69·3	59·3	SSW
31	29·562	29·592	29·651	29·783	64·3	56·1	SSW

1	29·855	29·925	30·028	30·091	67·1	55·9	NNE
2	30·044	30·068	30·044	30·036	67·7	52·9	W
3	29·964	29·962	29·934	29·968	62·1	47·7	WNW
4	29·975	29·997	29·988	29·983	62·5	45·0	WNW
5	29·879	29·809	29·715	29·647	61·9	52·1	WSW
6	29·597	29·641	29·712	29·804	59·9	48·5	WNW
7	29·827	29·875	29·864	29·829	61·9	48·0	W
8	29·738	29·731	29·780	29·885	53·0	46·7	NE
9	29·946	30·036	30·074	30·137	62·8	43·8	N
10	30·150	30·148	30·120	30·088	62·7	42·9	WSW
11	29·980	29·963	29·946	29·950	62·1	51·4	WSW
12	29·892	29·832	29·711	29·607	64·7	55·0	SW
13	29·458	29·422	29·384	29·372	65·8	54·1	SW
14	29·375	29·436	29·436	29·337	63·2	54·0	SW
15	29·358	29·550	29·654	29·782	62·0	52·8	SW
16	29·826	29·887	29·857	29·850	63·0	51·2	SW
17	29·811	29·866	29·899	29·928	65·7	50·1	W
18	29·863	29·786	29·674	29·629	54·8	52·1	NE
19	29·610	29·669	29·699	29·797	64·2	48·1	W
20	29·841	29·883	29·879	29·871	66·2	45·4	SSW
21	29·840	29·907	29·919	29·947	69·8	59·8	SW
22	29·909	29·895	29·840	29·809	67·1	59·7	SW
23	29·743	29·737	29·716	29·735	71·4	57·0	W
24	29·707	29·674	29·629	29·640	70·0	55·7	E
25	29·621	29·630	29·639	29·692	62·7	51·0	W
26	29·736	29·754	29·638	29·616	59·9	50·8	SW
27	29·499	29·440	29·401	29·401	60·0	49·0	SW
28	29·388	29·443	29·517	29·610	58·9	44·7	WNW
29	29·640	29·521	29·501	29·414	55·4	38·0	E
30	29·282	29·244	29·171	29·141	59·9	49·9	E
31	29·154	29·261	29·377	29·524	59·9	49·9	N

SEPTEMBER, 1892.

KEW.								GLASGOW.							
	Barometer.				Temperature.		Direction of wind at noon		Barometer.				Temperature.		Direction of wind at noon
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.		Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	
1	29·852	29·931	29·913	29·911	62·9	51·1	SW	1	29·539	29·453	29·469	29·441	56·2	47·0	W
2	29·767	29·650	29·604	29·623	63·0	52·3	SSW	2	29·338	29·324	29·308	29·344	54·7	45·2	WSW
3	29·639	29·670	29·704	29·824	60·7	47·8	W	3	29·382	29·480	29·558	29·834	57·5	45·3	WNW
4	29·909	30·044	30·143	30·259	58·5	48·0	NNW	4	29·976	30·102	30·133	30·190	56·5	41·9	SSW
5	30·294	30·339	30·310	30·325	61·7	39·7	SW	5	30·172	30·148	30·082	30·063	60·9	40·9	SW
6	30·291	30·289	30·220	30·208	63·3	42·9	SW	6	30·044	30·086	30·052	29·998	62·0	49·8	W
7	30·145	30·080	30·001	30·038	59·2	46·9	SSW	7	29·834	29·814	29·870	29·956	56·0	44·8	WSW
8	30·043	30·048	30·004	30·008	61·9	43·1	NNW	8	30·000	30·009	29·967	29·939	58·9	41·6	SW
9	29·971	29·959	29·938	29·969	62·0	42·4	NW	9	29·872	29·841	29·835	29·799	58·0	49·8	W
10	29·960	30·021	30·025	30·058	64·3	55·2	SW	10	29·829	29·889	29·925	29·915	59·9	48·9	W
11	30·068	30·092	30·028	30·077	65·4	52·9	SW	11	29·849	29·803	29·755	29·805	58·1	47·2	SW
12	30·087	30·103	29·969	29·841	68·9	58·4	S	12	29·811	29·792	29·662	29·560	59·0	52·3	SW
13	29·757	29·770	29·770	29·965	68·4	54·8	SW	13	29·432	29·446	29·696	29·848	58·1	46·0	W
14	30·067	30·133	30·109	30·128	65·5	46·0	W	14	29·870	29·876	29·837	29·782	55·2	46·1	SW
15	30·091	30·047	29·922	29·830	65·9	46·9	S	15	29·755	29·724	29·614	29·504	59·0	52·5	SSW
16	29·716	29·727	29·829	29·969	66·6	49·8	SW	16	29·322	29·333	29·423	29·637	56·7	48·0	SW
17	30·049	30·192	30·222	30·264	59·2	41·1	NW	17	29·872	29·986	29·972	29·918	56·0	41·5	SW
18	30·237	30·205	30·088	30·055	63·1	33·9	S	18	29·826	29·780	29·710	29·612	56·1	50·3	SW
19	29·981	29·997	29·963	29·982	70·9	46·9	SW	19	29·598	29·688	29·778	29·893	57·2	48·0	WSW
20	29·946	29·914	29·843	29·901	67·9	52·0	E	20	29·959	30·063	30·096	30·151	52·0	46·1	NE
21	29·918	29·944	29·944	30·049	66·6	55·0	N	21	30·170	30·236	30·237	30·283	52·0	40·0	ENE
22	29·105	30·166	30·167	30·173	58·9	55·1	NNE	22	30·291	30·295	30·235	30·206	55·0	39·2	ENE
23	30·113	30·097	30·036	29·984	63·5	53·8	WNW	23	30·103	30·008	29·843	29·694	55·5	41·4	WNW
24	29·877	29·889	29·922	29·983	64·2	48·0	W	24	29·552	29·624	29·651	29·568	54·0	46·0	W
25	29·947	29·943	29·901	29·974	63·0	46·6	SW	25	29·473	29·458	29·548	29·656	56·7	48·1	WSW
26	29·997	30·022	29·911	29·839	64·6	50·0	SW	26	29·688	29·702	29·566	29·394	57·1	48·8	SSW
27	29·752	29·708	29·632	29·591	66·3	51·9	SW	27	29·292	29·193	29·217	29·224	56·3	42·8	SW
28	29·679	29·760	29·734	29·773	58·1	46·2	W	28	29·404	29·528	29·500	29·415	51·2	40·9	W
29	29·796	29·784	29·649	29·591	58·3	42·6	SW	29	29·259	29·242	29·228	29·264	50·0	43·0	SW
30	29·473	29·457	29·500	29·480	57·1	47·8	SW	30	29·230	29·240	29·244	29·270	49·4	42·0	S

OCTOBER, 1892.

1	29·404	29·364	29·355	29·399	55·2	42·6	SSW	1	29·271	29·284	29·271	29·296	48·8	42·4	NE
2	29·446	29·440	29·487	29·594	49·3	39·3	ESE	2	29·350	29·414	29·479	29·569	47·8	41·9	NE
3	29·658	29·687	29·680	29·704	55·7	41·9	WSW	3	29·641	29·675	29·662	29·625	50·1	41·4	NNW
4	29·671	29·670	29·608	29·510	56·3	43·0	SW	4	29·581	29·598	29·567	29·538	51·8	46·0	W
5	29·420	29·382	29·370	29·437	55·1	40·3	SW	5	29·388	29·249	29·120	29·068	49·3	43·9	W
6	29·373	29·331	29·249	29·264	52·9	39·5	SW	6	29·113	29·158	29·150	29·169	51·7	46·0	SSW
7	29·276	29·341	29·355	29·404	55·2	42·8	SW	7	29·155	29·196	29·226	29·309	50·0	43·7	NW
8	29·427	29·534	29·572	29·608	51·6	42·1	WSW	8	29·338	29·403	29·373	29·226	48·2	40·2	WNW
9	29·497	29·370	29·410	29·494	53·5	42·8	W	9	28·937	28·851	29·043	29 290	49·2	42·1	WNW
10	29·605	29·754	29·828	29·934	55·8	39·2	W	10	29·479	29·622	29·724	29·848	51·7	38·0	NW
11	29·980	30·033	30·025	30·047	52·5	35·6	NW	11	29·915	30·015	30·052	30·109	52·9	36·3	NW
12	30·044	30·044	29·984	29·965	50·5	34·6	N	12	30·118	30·159	30·145	30·176	51·0	33·0	N
13	29·913	29·887	29·806	29·744	53·3	45·5	NNE	13	30·137	30·145	30·090	30·091	50·0	45·0	NNE
14	29·582	29·626	29·661	29·754	52·3	46·7	ESE	14	30·069	30·073	30·015	30·017	51·2	45·8	NE
15	29·783	29·804	29·748	29·744	52·0	40·1	E	15	29·991	30·023	29·994	30·007	48·0	42·7	NE
16	29·686	29·676	29·696	29·780	50·7	41·8	N	16	29·956	29·939	29·970	30·030	47·3	38·0	NE
17	29·826	29·944	30·023	30·108	48·9	37·5	N	17	30 057	30·120	30·151	30·234	45·6	35·0	NNE
18	30·144	30·209	30·224	30·274	48·0	35·5	N	18	30·272	30·310	30·285	30·274	46·9	30·1	W
19	30·271	30·288	30·211	30·198	47·9	32·9	NNW	19	30·209	30·135	30·068	30·092	49·1	34·8	W
20	30·132	30·081	29·995	29·932	48·0	35·5	W	20	30·027	29·969	29·873	29·767	45·0	37·8	ENE
21	29·795	29·665	29·540	29·512	49·7	38·6	W	21	29·653	29·632	29·611	29·679	44·4	36·3	N
22	29·494	29·586	29·611	29·598	46·0	34·7	NW	22	29·668	29·622	29·532	29·496	43·2	34·0	WNW
23	29·558	29·591	29·611	29·693	47·7	33·5	WNW	23	29·485	29·530	29·536	29·588	45·1	34·7	NW
24	29·711	29·752	29·782	29·834	46·3	31·0	W	24	29·679	29·755	29·783	29·826	42·1	29·3	NW
25	29·737	29·691	29·707	29·825	42·5	35·2	NNE	25	29·821	29·842	29·843	29·912	38·0	23·8	SSE
26	29·971	30·081	30·030	29·930	48·2	28·3	N	26	29·935	29·942	29·858	29·665	40·8	24·0	S
27	29·741	29·592	29·453	29·372	56·9	41·6	S	27	29·399	29·259	29·185	29·065	50·0	37·9	S
28	29·316	29·315	29·258	29·332	58·5	54·7	S	28	28·882	28·991	29·007	29·032	55·2	48·0	SW
29	29·328	29·349	29·442	29·543	58·9	46·3	SW	29	28·973	28·926	29·035	29·303	53·2	42·7	SSW
30	29·608	29·695	29·664	29·578	53·0	38·3	N	30	29·429	29·524	29·555	29·599	48·2	40·0	SW
31	29·480	29·607	29·679	29·744	47·0	45·1	N	31	29·591	29 643	29·690	29·762	46·3	35·7	WNW

NOVEMBER, 1892.

KEW.								GLASGOW.							
	BAROMETER.				TEMPERATURE.				BAROMETER.				TEMPERATURE.		
Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon	Date.	4 A.M.	10 A.M.	4 P.M.	10 P.M.	Max	Min.	Direction of wind at noon
1	29·765	29·816	29·822	29·852	48·9	34·9	N	1	29·792	29·833	29·810	29·798	44·3	30·4	NE
2	29·848	29·832	29·694	29·605	50·7	30·0	N	2	29·711	29·614	29·427	29·392	44·3	35·9	ESE
3	29·607	29·674	29·787	29·896	54·7	41·9	W	3	29·294	29·347	29·493	29·596	50·4	44·9	SW
4	29·878	29·862	29·817	29·792	58·0	42·2	S	4	29·482	29·415	29·446	29·484	55·0	47·6	SSW
5	29·831	29·873	29·839	29·839	57·2	50·5	SSW	5	29·456	29·487	29·580	29·668	52·1	45·8	SW
6	29·813	29·855	29·899	29·896	53·3	40·3	W	6	29·695	29·788	29·851	29·948	48·0	38·9	W
7	30·044	30·133	30·161	30·242	49·8	32·9	WNW	7	30·014	30·082	30·068	30·052	47·8	33·8	ENE
8	30·277	30·321	30·282	30·277	43·9	33·9	N	8	29·982	29·961	29·890	29·884	50·1	44·7	SSW
9	30·221	30·179	30·129	30·122	49·4	43·3	SE	9	29·859	29·857	29·881	29·951	51·0	41·9	SW
10	30·114	30·148	30·131	30·146	48·9	37·0	SE	10	29·964	29·996	29·959	29·976	48·3	38·9	SW
11	30·131	30·120	30·072	30·048	47·9	36·3	W	11	29·930	29·936	29·931	29·951	52·8	44·7	SW
12	30·000	29·981	29·915	29·905	50·4	42·2	SSW	12	29·903	29·849	29·738	29·657	48·5	43·3	S
13	29·841	29·825	29·776	29·789	54·9	44·1	S	13	29·617	29·610	29·577	29·617	47·8	42·9	WSW
14	29·785	29·795	29·728	29·689	60·1	51·1	S	14	29·599	29·531	29·349	29·200	54·9	42·7	ENE
15	29·664	29·727	29·756	29·762	56·7	49·9	W	15	29·201	29·398	29·576	29·689	50·9	40·2	WSW
16	29·687	29·755	29·790	29·880	54·0	47·0	N	16	29·710	29·775	29·797	29·879	43·2	34·0	NE
17	29·953	30·006	29·977	29·984	47·2	38·8	N	17	29·930	29·939	29·897	29·880	45·3	35·8	SE
18	29·951	29·932	29·840	29·783	43·4	42·9	SW	18	29·811	29·796	29·728	29·685	43·3	36·2	ENE
19	29·692	29·689	29·694	29·749	49·7	36·0	SE	19	29·612	29·624	29·664	29·742	40·7	34·0	E
20	29·817	29·991	30·097	30·176	45·9	33·8	WSW	20	29·814	29·929	30·010	30·143	46·1	40·0	ENE
21	30·224	30·307	30·302	30·355	45·1	34·5	WSW	21	30·208	30·305	30·314	30·351	44·7	42·0	ENE
22	30·340	30·349	30·302	30·291	44·4	41·7	E	22	30·334	30·367	30·335	30·331	43·3	38·0	ENE
23	30·267	30·289	30·254	30·242	44·5	39·4	E	23	30·297	30·291	30·222	30·190	42·0	36·6	ENE
24	30·191	30·195	30·147	30·168	43·5	38·4	SE	24	30·122	30·131	30·140	30·158	39·3	37·8	ENE
25	30·168	30·198	30·165	30·179	44·7	35·3	E	25	30·146	30·159	30·094	30·031	43·0	38·1	E
26	30·178	30·158	30·060	30·141	50·1	40·9	S	26	29·899	29·806	29·933	30·100	46·0	39·8	NW
27	30·219	30·372	30·419	30·456	48·9	40·1	N	27	30·150	30·154	30·161	30·151	50·1	42·1	WSW
28	30·432	30·436	30·348	30·260	48·0	42·7	W	28	30·095	30·014	29·853	29·617	52·0	47·4	SW
29	30·160	30·086	30·014	30·009	52·4	41·7	SW	29	29·671	29·744	29·632	29·662	49·0	33·3	WSW
30	30·029	30·140	30·198	30·217	43·2	38·8	WNW	30	29·760	29·980	29·976	29·926	37·6	32·3	W

DECEMBER, 1892.

1	30·111	30·000	29·674	29·795	50·1	34·7	SW	1	29·816	29·756	29·790	29·892	36·0	24·9	ENE
2	29·998	30·154	30·129	30 036	39·5	32·2	N	2	30·010	30·110	30·049	29·903	27·0	19·0	ENE
3	29·784	29·628	29·522	29·511	50·7	37·1	SW	3	29·612	29·349	29·257	29·267	39·4	23·9	WSW
4	29·523	29·603	29·664	29·710	39·8	31·2	WNW	4	29·311	29·457	29·483	29·545	31·5	26·5	W
5	29·672	29·713	29·683	29·738	36·2	29·9	WNW	5	29·589	29·578	29·524	29·604	34·8	26·5	NW
6	29·757	29·827	29·854	29·846	39·0	28·8	W	6	29·643	29·677	29·816	29·949	40·8	33·8	NW
7	29·878	29·983	30·060	30·137	39·9	34·5	N	7	30·061	30·166	30·177	30·215	38·1	31·2	NNE
8	30·189	30·237	30·152	29·989	38·1	32·7	WNW	8	30·214	30·132	29·888	29·652	36·0	28·0	NE
9	29·555	29·606	29·711	29·835	41·4	32·8	N	9	29·765	29·888	29·953	29·991	35·2	27·0	NW
10	29·918	30·007	29·935	29·738	40·7	31·9	NNW	10	29·923	29·773	29·467	29·273	41·9	24·8	SW
11	29·503	29·434	29·404	29·459	45·3	40·1	W	11	29·148	29·083	29·104	29·224	41·6	36·0	WSW
12	29·511	29·523	29·492	29·647	43·9	35·3	SW	12	29·271	29·343	29·443	29·519	39·3	33·2	WNW
13	29·749	29·921	30·051	30·153	41·1	31·1	NW	13	29·704	29·872	29·928	29·892	37·8	32·0	W
14	30·150	30·059	29·961	29·993	50·1	31·3	SW	14	29·686	29·648	29·722	29·801	46·3	36·9	WNW
15	30·019	30·622	30·029	30·172	54·0	47·0	W	15	29·782	29·734	29·829	29·989	45·2	42·7	WSW
16	30·299	30·339	30·299	30·290	49·9	36·0	SW	16	30·090	30·033	29·887	29·827	48·8	39·1	SSW
17	30·279	30·300	30·308	30·315	49·1	47·1	WSW	17	29·859	29·952	29·956	29·886	51·0	46·9	W
18	30·273	30·277	30·239	30·253	48·5	46·6	WSW	18	29·776	29·883	29·899	29·932	50·9	45·8	W
19	30·215	30·199	30·122	30·109	46·6	42·1	WSW	19	29·946	29·952	29·939	29·946	47·8	45·0	NW
20	30·063	30·084	30·054	30·077	43·6	41·2	W	20	29·953	30·015	30·014	30·017	45·0	38·8	ENE
21	30·072	30·098	30·079	30·087	41·5	31·5	NNW	21	30·011	30·012	29·995	29·999	41·1	39·2	ENE
22	30·060	30·045	30·011	30·030	40·8	31·3	ENE	22	29·978	29·993	29·992	30·025	41·5	38·9	E
23	30·029	30·073	30·042	30·038	38·9	30·9	ESE	23	30·036	30·074	30·035	30·018	39·1	30·9	ESE
24	29·998	29·997	29·960	29·958	33·3	27·1	ESE	24	29·962	29·957	29·912	29·886	31·6	26·7	E
25	29·893	29·902	29·909	29·974	32·9	26·4	E	25	29·861	29·892	29·928	30·011	26·7	20·0	NE
26	30 036	30·106	30·122	30·161	33·7	22·5	E	26	30·042	30·090	30·102	30·145	30·0	17·2	WSW
27	30·173	30·242	30·238	30·307	32·2	17·9	E	27	30·149	30·197	30·192	30·215	31·0	19·3	WNW
28	30·299	30·325	30 256	30·195	28·3	24·4	E	28	30·173	30·149	30·041	29·987	38·7	28·5	SW
29	30·085	30·020	29·935	29·894	33·5	20·3	E	29	29·864	29·801	29·743	29·726	40·1	37·0	S
30	29·861	29·849	29·813	29·802	32·2	19·9	SE	30	29·720	29·723	29·727	29·753	38·7	30·5	E
31	29·786	29·805	29 799	29·853	34·6	21·9	E	31	29·816	29·854	29·874	29·916	36·1	26·9	E

II.—NOTES OF PAPERS ON THE WORKING OF MINES, METALLURGY, ETC., FROM THE TRANSACTIONS OF FOREIGN SOCIETIES AND FOREIGN PUBLICATIONS.

ALUMINIUM IN THE UNITED STATES.

Aluminium. *By* W. R. Ingalls. *Engineering and Mining Journal* (*New York*), 1892, *vol. liii.*, *pages* 5-6.

Bauxite occurs in Tennessee, Virginia, North and South Carolina, Georgia, Alabama, and Arkansas. Its discovery in the latter State was only announced in the early part of 1891 by Dr. J. C. Branner, State Geologist.* Bauxite, used in the manufacture of aluminium, is worth about £2 per ton at Pittsburg.

Cryolite is used in the manufacture of aluminium by the electric process. It is imported from Evigtok, near Arksut, Greenland, that being the only country in which the mineral is mined. The Evigtok mine is worked as a quarry 450 feet long, 150 feet wide, and 100 feet deep, while diamond drills have proved its existence to a depth of 150 feet deeper. The only place in the United States where cryolite is known to exist is at Pike's Peak, Colorado, but no workable deposits have yet been found.

Aluminium is generally produced by one of the electric smelting methods (the Cowles, Hall, Heroult, or Minet), which have almost entirely superseded the old sodium process. The bauxite is generally calcined with enough sodium carbonate to form sodium aluminate, and the incinerated mass is ground and leached with hot water, which dissolves the aluminate, leaving the silica and iron behind. The solution of sodium aluminate ($Na_2Al_2O_4$) is filtered, and the alumina (Al_2O_3) precipitated by carbonic acid gas is filtered, washed, and dried. The alumina is next dissolved in molten cryolite ($Al_2F_6 + 6NaF$) in pots and electrolysed, the apparatus and the details of the operation varying with the different processes.

J. W.

AMBER IN BURMA.

On the occurrence of Burmite, a new fossil Resin, from Upper Burma. *By* Dr. Fritz Noetling. *Records of the Geological Survey of India*, 1893, *vol. xxvi.*, *pages* 31-40.

The amber of Burma, known as burmite, differs from ordinary amber (succinite) by containing no succinic acid. It occurs abundantly, but is of less value than common amber, on account of its colour and the occurrence in it of fissures filled with calcspar.

G. W. B.

AMBER AND JADE IN UPPER BURMA.

Preliminary Report on the Economic Resources of the Amber and Jade Mines Area in Upper Burma. *By* Dr. Fritz Noetling. *Records of the Geological Survey of India*, 1892, *vol. xxv.*, *pages* 130-135.

The amber occurs in a soft blue clay, probably of Lower Miocene age. It is found in lumps of various sizes up to that of a man's head, and rounded, or flattened, in shape. The colour varies from light yellow to dark brown, and it is fluorescent.

**Trans. Fed. Inst.*, vol. iii., page 1057.

The jade mines are "pit mines" and "quarry mines." The former are situated on the banks of the river Uru, and the latter near the village of Tammaw. In the Tammaw mines the jade occurs as a vein in a blackish-green igneous rock. It is described as a "purely white cryptocrystalline mineral, much resembling the finest marble, containing here and there green specks of various sizes, which form the jade proper."

In the pit mines it occurs in boulders derived from other localities.

At least 500 men are engaged every season in working the quarry mines at Tammaw. The rock is heated by large fires, and, after cooling, is broken in pieces by hammers.

In the pit mines the miners simply dig a pit, and select boulders from the stuff dug out. Good jade is also sometimes found in the laterite.

The Report also contains a note on the Mogaung coal-field in the same area, which is of Tertiary age. G. W. B.

ASBESTOS.

The Chemical Composition of Asbestos. *By* J. T. DONALD. *Engineering and Mining Journal* (*New York*), 1891, *vol. li., page* 741.

The following analyses show the composition of Italian and Canadian asbestos :—

	Italy.	Canada. Broughton.	Canada. Templeton.
Silica	40·30	40·57	40·52
Magnesia	43·37	41·50	42·05
Ferrous oxide... ...	·87	2·81	1·97
Alumina	2·27	·90	2·10
Water...	13·72	13·55	13·46
	100·53	99·33	100·10

Chemical analysis shows that the cause of the harshness of fibre of some varieties of asbestos is due to loss of water. J. W.

ASBESTOS MINING IN QUEBEC.

By R. W. ELLS. *Engineering and Mining Journal* (*New York*), 1891, *vol. li., page* 498.

The asbestos mines of Quebec are practically the only deposits whose mineral is of a quality adapted for spinning and for the finer purposes of manufacture.

The asbestos veins are associated with rocks which for thirty years have been known as the Quebec Group. They comprise an extensive series of both sedimentary and igneous rocks, which extend from the Vermont boundary to the extremity of the Gaspé peninsula. Crossing the Gulf of St. Lawrence, the rocks also occur in Newfoundland, where specimens of asbestos have been found in course of geological explorations, although no systematic search has been made for the mineral.

Asbestos proper belongs to the hornblende or pyroxene (*sic*) group of minerals; while that of Quebec, commonly called asbestos, is really a variety of serpentine, known mineralogically as chrysotile. Sometimes these masses of serpentine assume such proportions as to rank almost as mountain-ridges.

Up to 1880, most of the fine asbestos fibre, for spinning, came from Italy and Corsica; it was hard to obtain, and of exceptionally fine quality, commanding as

high a price as £60 per ton. The discovery of the Quebec chrysotile, quite as well adapted for spinning as the Italian asbestos, almost revolutionized the industry, and has nearly closed the Italian mines.

So much of the serpentine of Quebec as is really asbestos-bearing to an extent which can be profitably worked, is confined to a comparatively limited area—in the townships of Thetford, Ireland, Coleraine, and Wolfestown, where successful mining operations have been carried on for some years.

The veins of asbestos range from mere threads up to a thickness of 5 or 6 inches, though most of the workable veins do not exceed 2½ inches in width or length of fibre, and such veins of good quality and unbroken by partings of iron are regarded as extra No. 1 material. The asbestos veins occur throughout the mass of rock, and crop out where exposed; the mining is therefore simply open-quarry work, the entire rock being removed, broken up, and the veins of asbestos separated by hand-cobbing.

It has been estimated that the bulk of barren serpentine removed to obtain a ton of fibre varies from 25 to 1 on prolific ground to 50 to 1 on ordinary ground.

Little attention was paid to the occurrence of asbestos in Canada up to 1877, when the burning of the forests in Thetford and Coleraine townships laid bare the hills of serpentine. A French-Canadian named Fecteau observed the felting of the weathered asbestos, and the importance of the new material resulted in the operations of Ward Brothers, and later of the Johnson Asbestos Mining Company.

The output has increased from 50 tons in 1878 to about 8,000 tons in 1890. Sufficient study of these peculiar rocks has not yet been made to pronounce definitely upon their probable importance; but, when the deposits are made more accessible, considerable mining will undoubtedly be done, as these appear to be quite extensive. J. W.

JOINTED EARTH-AUGER FOR GEOLOGICAL EXPLORATIONS IN SOFT DEPOSITS.

By N. H. Darton. *The Engineering and Mining Journal* (*New York*), 1891, *vol. li., page* 401.

This auger consists of an auger-bit, a number of three-feet lengths of ¼ inch iron pipe, a cross-bar handle, all with threads and couplings for connexion to any length required; a pair of pipe tongs, and a receptacle of some kind. An ordinary 1½ inches carpenter's auger, with two or three whorls of double spirals, forms the bit; this is welded to a ½ inch iron bar to extend its length to three feet. The labour of working is not great, if care be taken not to bore too deeply at a time, and one man can make good progress in soft materials.

The instrument is said to be simple and inexpensive, and likely to prove useful in searching soft deposits at depths not exceeding 40 feet. J. W.

ON THE EXTENSION OF THE SAARBRÜCKEN COAL-FIELD INTO BAVARIA.

Ueber die Fortsetzung des Saarbrücker productiver Steinkohlengebirges in der Bayerischen Pfalz. By M. Kliver. *Zeitschrift für das Berg-Hütten-und Salinen-Wesen, vol. xl., page* 471, *and plate* XVII.

The author discusses the probability of finding the Saarbrück coal formation in Bavaria. Elaborate details are given of the various coal-seams found by the boring experiments. W. F. W.

COAL-MINING IN HOLLAND.

Der Bergbau im Kœnigreich der Niederlande. *By* FRANZ BÜTTGENBACH. *Berg- und Huettenmœnnische Zeitung, vol. li., pages* 221-223.

The Dutch Government have granted concessions for the working of recently-discovered extensive coal-fields in the Province of Limburg, on condition that operations are commenced within a short period of time, and it may be expected that coal from this district will shortly take a prominent place in the market.

The Worm valley, on the frontier between Prussia and Holland, lies chiefly in the former country, but a small portion of it extends into Dutch territory, and contains the two coal-mines of Neuprik and Bleyerheide, now consolidated into one, and a third, called the Domanial mine, which is under Government control. The latter extends partly into Prussian territory, and by a singular treaty (concluded in 1816) may be worked across the frontier without interference from the Prussian mining authorities.

The coal is anthracitic, containing about 90 per cent. of carbon and 2·3 per cent. of ash, and yielding about 48 per cent. of large coal. It is excellent for household purposes, being very free from soot and smoke. The small coal is mixed with clay, and much used in stoves.

The Government mine is now let to the Aachen-Maastricht Railway Company, who raise about 200 tons per day, and employ 310 men.

The united Neuprik and Bleyerheide mine lies on Dutch territory, but is worked by a Prussian company. It is connected by a drift with the Neuvoccart mine across the frontier. At the frontier the drift is fitted with iron gates, locked, sealed, and placed under control of the Custom House officials.

The coal-mining district, with an area of 1,450 acres, is situated at and around the town of Limburg.

In addition to the coal-mines, Holland has at work, near Valkenburg and Maastricht, nine underground mines for building-stone, including the well-known one of St. Pietersberg.

Interesting particulars are given about the Miners' Mutual Aid Society, which is in a flourishing condition. The society grants pensions to those members who are over fifty years of age, and who have contributed to the funds for five years. The contributions amount to 3 per cent. of the wages. The society provides liberally in cases of sickness or accident, and the children of members receive free education. Contributions are regulated according to the time of membership and according to classes. The employer pays half the amount contributed by the workmen. The society has existed thirty-four years, and possesses funds to the amount of £1,168. The miners are also provided with good houses, and support co-operative stores.

A. R. L. AND W. F. W.

MARINE BEDS IN THE CRACOW COAL-MEASURES.

Ueber eine marine Einlagerung im productiven Carbon der Krakauer Gegend. *By* DR. E. TIETZE. *Verhandlungen der k. k. geologischen Reichsanstalt,* 1892, *pages* 76-77.

In 1891, two new shafts were sunk in the immediate neighbourhood of Tenczynek (mining district of Cracow); and, at a depth barely exceeding 42 feet from bank, a seam of gas coal, about 2 feet thick, was struck. The roof of the seam is formed by a band, about $4\frac{1}{4}$ inches thick, of bituminous shale, crowded with fossils; these, on examination, were found to consist entirely of the remains of *Lingula squami-*

formis, Phil. The same genus of brachiopod occurs in the Upper Silesian coal-field, and at Ostrau, in conjunction with a rich marine fauna ; but in those localities the genus *Lingula* is not represented in sufficient abundance to make its occurrence characteristic.

The find at Tenczynek appears to show that the Cracow Coal-measures are at a lower horizon than was heretofore generally suspected. It proves also the extension of marine conditions very nearly up to what is the presumed easterly limit of the coal-field. O. S. E.

THE DONETZ COAL-FIELD, RUSSIA.

Die Kohlenlager im Donetzki-Bassin in Russland. By N. Syrkin. *Berg-und Huettenmænnische Zeitung, vol. li., pages* 203-204 *and* 241-243.

Russia produced 3½ million tons of coal in the year 1880, *i.e.*, less than 1 per cent. of the output of the world. In 1886, the output rose to about 5 million tons, and it is now still greater, but the production per head of the population remains less than one-sixth of that for the rest of the world.

At the same time scarcely any other country is so rich in coal. In Saghalien valuable seams occur, which, though worked in a small way, are too far from European centres of industry to become of importance. Near Kusnezk, in Western Siberia, thick seams of good quality lie unworked near the surface, timber being abundant in the neighbourhood and no centre of industry near. On both slopes of the Urals large coal-fields are known, but, although railways and rivers afford good facilities for transport, mining has made little progress there also.

The Khirghiz steppes and the Turkestan district are likewise rich, and geologists affirm that beds of coal stretch from the White Sea to the Sea of Azov. The coal-seams are mostly at considerable depths, but in many cases they are near the surface, and conveniently situated for working and exportation. Coal of rather poor quality occurs also in Poland, and is there extensively worked. The Caucasus Mountains are very rich in coal, and at Tkwibula, near Kutais, seams more than 15 feet thick lie near the surface, and within easy reach of the railway and the sea.

The richest of all the Russian coal-fields is the Donetz basin, which may some day come into active competition with England in the Mediterranean. It has a known area of 11,000 square miles, and future exploration may extend its limits still further. It has numerous seams lying near the surface and within depths of not more than 600 feet. These contain coal enough to supply the wants of the world at the present rates of consumption for fifty years to come, while the supplies at lower depths are doubtless much greater. In the shallow mines, which have hitherto been worked, explosive gas is unknown, and ventilation either by furnace or fan is unnecessary.

Anthracite occurs in the south and east of the district, smithy and coking coals in the west, and gas coal in the north, the latter being mostly used on the railways.

The district is considered by Prof. Mendelejew to be the richest in the world. The coal contains about 7 to 8 per cent. of ash, and sometimes as little as 2 to 3 per cent., with not more sulphur than English, and much less than Westphalian and Silesian coals. It contains little moisture, and when lying in heaps will not take fire spontaneously or crumble to small pieces. It can be worked at less cost than English or any other coal except that of Silesia, and, with the introduction of modern methods of working and a general development of industry, it may be expected to become cheaper in price.

The district is traversed by the Kursk-Charkov-Azov, the Woronesh-Rostov, the Ikaterinin, the Rostov-Wladikavkass, and Don railways, the last-mentioned, called also the coal railway, having branch lines over the whole district. In addition, the Donetz river, flowing for 400 miles through the richest coal country, admits of improvement for the passage of ships. A. R. L.

THE HERAKLEA COAL-FIELD, TURKEY IN ASIA.

Notice sur le Bassin Houiller d'Héraclée (Turquie d'Asie). *By* J. D'HARVENG. *Publications de la Société des Ingénieurs, etc., du Hainaut, third series, vol. i., pages* 278-314, *and two plates.*

The Heraklea coal-field extends along the shore of the Black Sea from Armutchuk, which is 7½ miles east of Heraklea, for a distance not yet exactly known. The author studied the general outline of the basin to a distance of 27 miles east of Armutchuk, or about 20 miles of the coal-field. The extent of the basin is probably at least 90 miles, and to the east it recedes from the sea. On the shore its known width at present does not exceed 1½ miles.

Twenty-five seams are worked having a united thickness of coal of 138 feet, or an average of 5½ feet, but the total number of workable seams is 42, of an average thickness of 3·38 feet. The coal-field is contorted, causing great variations in the inclination of the seams.

The existence of this coal-field has been known, and it has been worked to a very limited extent, from ancient times. Some impulse was given to its working during the Crimean War by the importation of miners to coal the British fleet. Since then working has been more extensive than formerly, but is still on a very limited scale, as the supply of labour appears to be insufficient. The natives have few wants and only require money to pay taxes, and for this purpose they are obliged by the Government to work in the mines, but during autumn and winter the supply of labour is irregular.

The method of working is most simple. When the seam is flat it is cut from the outcrop by two drifts 70 to 100 yards apart at different levels, so as to establish a ventilation; between these roads pillars are formed 20 to 35 yards square, until some slight difficulty arises, and then the pillars are drawn back and completely removed. In steep seams the same principle is applied, only the length of the pillar is reduced to 4 or 5 yards.

There are two collieries worked by shafts about 45 yards deep, and five by slopes with steam winches. The coal is carried from the working-places in panniers to the haulage-road, where it is emptied into tubs holding 2½ cwts., which are pushed to the outlet, and the coals tipped over screens.

The coal is taken from the mines by rail to the sea-shore and discharged at a wharf into caïques, which convey it to vessels, which cannot approach nearer than 330 yards to the shore. There is as yet no port whatever, but it has been proposed to form one at Cozlon or Yunguldak.

The coal-field of Heraklea is one of the sources of the civil list of the Sultan, and is under the charge of the Marine Minister. Concessions are readily granted, but only to Turkish subjects, a difficulty always easily surmounted by foreigners. Concessions are granted for ninety-nine years, subject to liability to dispossession on payment of compensation. A concession usually comprises a radius of 500 yards from a fixed point, and applies to one seam only. The administration of the civil list reserves all the small coal produced, and sells it. The coal being usually rather hard, the proportion of small does not exceed 25 per cent. On the rest of the

production 1s. 8d. per ton is paid. It is also provided that in case of pressing need, as in time of war, the Government may appropriate 60 per cent. of the large coal at the price of production.

The total production of the coal-field in 1890 was 76,000 tons, and in 1891 about 90,000 tons. The number of mines is about 105, but many of them produce little or no coal. Only one mine can produce 100 tons per day, and others produce 12,000, 10,000, 6,000, etc., tons per year.

As to consumers, the Government takes from 35,000 to 40,000 tons. Other consumers are the Ottoman and Balkan railways, steamboat companies, Constantinople and other towns. The coal is too bituminous to be good steam coal, but is used for that purpose because it is cheaper than Cardiff coal. W. N. A.

TENASSERIM COAL-FIELD, INDIA.

Coal on the Great Tenasserim River, Mergui District, Lower Burma. By T. W. H. Hughes. *Records of the Geological Survey of India*, 1892, *vol. xxv., pages* 161–163.

This paper records the re-discovery of coal on the Great Tenasserim river, first announced by Dr. Oldham in 1855.

A seam cropping out on the banks of a tributary of this river has the following section :—

	Ft.	In.
Coal (A)	0	10
Shale	2	0
Coal (B)	2	3
Shale	3	0
Coal (C)	4	6

Three samples gave the following analyses :—

Specimens.	A	B	C
Moisture ...	15·20 ...	10·80 ...	11·34
Volatile matter ...	30·08 ...	27·36 ...	36·40
Fixed carbon ...	30·86 ...	42·52 ...	43·27
Ash ...	23·86 ...	19·32 ...	8·99
	100·00 ...	100·00 ...	100·00

Does not cake, and yields reddish brown ash.

Specimen C, from the bottom of the seam, is a hard jet-like variety, fitted to stand the wear and tear of transport, and containing little pyrites. The coal is above the average Indian coal, and when tested on the Government steam launch gave satisfactory results. The coal-field is unfortunately situated as regards labour and carriage, yet the writer estimates that on an output of 10,000 tons per annum it might be worked, and conveyed to Mergui, for something like 11s. per ton.

G. W. B.

DALTONGUNJ COAL-FIELD, INDIA.

Boring Exploration in the Daltongunj Coal-field, Palamow. By Tom D. La Touche. *Records of the Geological Survey of India*, 1891, *vol. xxiv., pages* 141-153, *and Map.*

The results are described of investigations of this coal-field by means of borings, made with the view of testing whether coal could be produced in sufficient quantities to warrant the construction of a railway, which would make it available for use in Upper India. It is concluded that it is extremely doubtful whether coal could be produced in sufficiently large quantities.

The coal-bearing rocks are of Karharbari age, and occupy an area of about 30 square miles.

Dr. Saise's estimate, of not less than 161,377,000 tons of coal with 11·7 per cent. of ash for this coal-field, is considered too favourable. Analysis shows that the coal contains more ash than is admissible in a serviceable coal. This, however, may be partly due to admixture with sand and clay during boring. G. W. B.

NORTHERN SHAN STATES COAL-FIELDS, INDIA.

Report on the Coal-fields in the Northern Shan States. *By* FRITZ NOETLING. *Records of the Geological Survey of India*, 1891, *vol. xxiv., pages* 99-119.

This paper is a preliminary description of certain coal-fields in the Northern Shan States. The coal is probably of late Tertiary age, but its exact position is uncertain. The seams vary from 2 inches to 30 feet in thickness, the average being 5 to 6 feet, the thin seams not being frequent. The following analyses are given :—

No. of Sample.	Description.	Volatile Matter and Moisture.	Fixed Carbon.	Ash.	Remarks.
1	Coal from Lashio, dull colour, very brittle	66·75	24·05	9·20	Burns readily, with reddish ash.
2	Coal from Lashio, probably different from No. 2 seam...	54·48	35·44	10·08	Burns readily, with reddish ash.
3	Coal from Naleng (Lashio coal-fields)	33·18	33·06	12·46	Does not cake, not with vapour even (*sic*); ash dark red, contains much iron pyrites; 21·3 per cent. of moisture.
4	Coal from Namma seam (No. 1), of bright colour, but rather friable	56·82	38·58	4·60	
5	Coal from Namma seam (No. 2), of bright colour, but rather brittle	52·59	37·64	9·77	Average, with intervening earthy matter.
6	Coal from Namma seam (No. 3), of bright, jetty colour, hard, but laminated; best sample	60·45	36·15	3·40	Fracture glistening.
7	Coal from Namma seam (No. 4), of bright colour, but rather brittle	58·37	33·47	8·16	
8	Coal from Namma seam (No. 5), of bright colour, laminated, and brittle	56·08	34·08	9·84	
9	Coal from Hoetan-paipa, on the Namma stream, of bright colour; friable ...	53·49	35·12	11·39	
10	Coal from Namsale, a tributary of the Nampong; dull-looking	53·71	31·69	14·60	
11	Coal from Manzè seam (No. 1), of bright colour, very brittle; 6 inches seam ...	53·60	34·22	12·18	
12	Coal from Manzè seam (No. 2), dull-looking, brittle; 3 inches seam	55·42	34·94	9·64	

These coals form a very good fuel when fresh, and being hard would bear transport.

The coal of the Northern Shan States is very much poorer than that of the southern, yet it is more favourably situated for working.

The following coal-fields are mentioned:—1. The Lashio coal-field; 2. The Manzè-Namma coal-field; 3. Theinnimyo; 4. Namyaw Valley, near the village of Meungyaw; 5. Mansang and Meungpat; 6. Nong Hswan in Kyithi-Bansan State; 7. Silan on the Nammaw or Shwely river. Only the first two are described.

The writer's conclusion is that these coal-fields will be of no value so long as there are no means of transport by which the coals can be easily brought down the Irrawaddi river. Superficial accumulations will make sinking troublesome, and the climate predisposes to fever. G. W. B.

DARJILING COAL-FIELD, INDIA

Further Note on the Darjiling Coal Exploration. *By* P. N. Bose. *Records of the Geological Survey of India*, 1891, *vol. xxiv., pages* 212-217.

The result of this exploration is that no promising coal-seams have been discovered. G. W. B.

PROSPECTS OF GOLD, COAL, AND TIN IN THE MERGUI DISTRICT, INDIA.

Report on the Prospecting Operations, Mergui District, 1891-92. *By* T. W. H. Hughes. *Records of the Geological Survey of India,* 1893, *vol. xxvi., pages* 40-53.

This paper records the results of prospecting for tin, coal, and gold, confirms previous reports, and states that the tin deposits are sufficiently large and accessible to be worked profitably under economical management. The geological structure of the country, moreover, gives reason to hope that, both north and east of the district prospected, further paying tin deposits may be found.

The Tenasserim coal-field is estimated to contain 1,000,000 tons. The estimated cost on a yearly output of 10,000 tons is about 8s. per ton, transit from the coal-field to Mergui 2s. 10d. per ton, total, 10s. 10d. per ton. Other markets besides the local markets would be Rangoon and Penang.

Little is mentioned as to gold.

Appendices I. and II. contain the reports of Mr. R. Ross Clunis and Mr. Alexander Primrose respectively. G. W. B.

COAL IN THE FAR EAST.

Le Charbon en Extrême-Orient. *By* G. Fontaine. *Revue Maritime et Coloniale,* 1892, *vol. cxv., pages* 539-559.

The various Eastern countries visited by French men-of-war possess important resources of coal, which are being utilized to an increasing extent. East Asiatic coal is, however, always inferior to the European coals, giving a dense smoke and more ash, and as the quantity which can be burnt per hour is limited, the maximum speed obtainable is two or three-tenths less than with European coal.

Particulars of many of these sources of supply are given as follows:—

Saghalien.

Several mines are worked, producing generally lignite of good quality. Recently bituminous coal from this island, giving a calorific value of 0·8 as compared with Welsh coal, was procurable at Vladivostock at 37s. 6d. per ton.

Japan.

Numerous coal-basins exist in the islands of Yeso, Nippon, Shkoku, and Kushiu, and a great number of mines are worked. At the Tokio exhibition in 1890, samples were shown from nine different provinces.

In Yeso there are four basins of bituminous coal, the contents of which have been estimated at 400,000,000 tons. Hakodaté is supplied by the Pororai mines, the product from which sells at 27s. as against 56s. per ton for Cardiff coal, compared with which its calorific value is 0·8. As sold it is of large size, is very flaming, and lights easily.

Takashima is the largest coal-mine in the island of the same name, which has an output of 900 tons per day. The following is an analysis of the coal:—

Volatile matter	39·00
Fixed carbon	55·60
Ash	5·40
	100·00

It contains 0·081 per cent. of sulphur, less than is usual with Japanese coal, and costs 24s. 6d. per ton delivered at Nagasaki. The principal objections to the use of this coal are that it has too much dust and makes much smoke.

China.

China is said to possess coal-fields of enormous extent; that of Sechuen, in the interior, covering 250,000 square miles, but the production is small, and the quality of the products brought to the coast bad.

Formosa.

The basin of Kelung, on the north coast of Formosa, covers an area of 5,000 acres, and contains four beds, varying in thickness from 20 inches to 4½ feet. The output in 1882, when sickness put an end to the working, was 74,000 tons per annum. The works have since then been re-established, and the coal in 1889 sold at 24s. per ton.

East Indian Archipelago.

Borneo, Java, Sumatra, Cebou and other islands contain coal, which, however, does not appear to be actively worked.

Tonkin, etc.

Coal exists in several places, and is worked at Kebao and Turane, as to the product from which the following details are given:—

	Volatile Matter. Per Cent.	Fixed Carbon. Per Cent.	Ash. Per Cent.
Kebao ...	33·6 to 36·6 ...	59·2 to 61·8 ...	4·7
Turane ...	6 to 15 ...	74·6 to 77 ...	8 to 19·4

Both are anthracitic coals, and the latter in particular will not support combustion by itself.

The paper concludes with elaborate comparisons as to useful effect between these Eastern coals and other well-known products. G. E. C.

EARTHQUAKES IN ALGERIA.

Note sur les tremblements de terre en Algérie. *By* G. CHESNEAU. *Annales des Mines*, 1892, *series* 9, *vol. i., pages* 5-46, *and plates I. and II.*

The author, a Government engineer, was apparently deputed by the French Ministry of Public Works in 1888 to study the question of earth-tremors in Algeria in connexion with the geological structure of the colony.

He gives a concise account of the shocks which have been recorded there between 1639 and 1888, the earthquake of 1790 being mentioned as especially disastrous. Of more recent earthquakes, those of 1867 and 1887 were the most violent.

It is pretty conclusively shown that the axiom laid down by Italian seismologists—viz., that in zones liable to earthquakes the direction of the strongest horizontal shocks is either perpendicular or parallel to the principal lines of fracture or plication in the strata of the country—is applicable to Algeria.

The character of the rocks is an important factor, both in the speed of propagation and in the dynamical effects of earth-tremors. Through thick deposits of drift or alluvium, for instance, the shock is transmitted with somewhat the same difficulty as a sound-wave through bran or sawdust. But a comparatively thin layer of sand or loam, resting on a more compact rock, behaves much as the vibrating sand-grains on a sounding-board.

Outcrops of crystalline and Palæozoic rocks are, comparatively speaking, seldom met with in Algeria; one notable exception is the patch of gneiss and schists on which the city of Algiers is built. A great band of Jurassic marls and limestones runs from west to east, partly concealed by newer strata. North and south of that strip appear the alternating clays and sandstones of the Gault, succeeded in their turn by Eocene, Miocene, Pliocene, and a vast extent of Quaternary deposits. The geological *résumé* is elucidated by a map which accompanies the original paper.

The author remarks in conclusion that most Algerian towns (excepting Algiers and Constantine) are, from the seismological standpoint, built on sites as ill-chosen as could possibly be. They stand indeed in the best cultivated districts of the dependency; now, these naturally irrigated, good arable lands are low-lying alluvial plains stretching along the base of mountain-ranges, forming just the kind of country which experience has shown to be the most exposed to seismic disturbances.

O. S. E.

ELECTRIC AUXILIARY HAULING-MOTOR.

By THOMSON VAN DEPOELE. *Electric Winding Engines for Mine Haulage. Engineering and Mining Journal (New York), vol. li., page* 659, *with illustration.*

In many cases where electricity is used for mine haulage it happens that there is only one steep grade on the line, without which it would be possible to use much smaller electric locomotives, or, it may be, after a plant has been installed, extension up a steeper grade is necessary. In such cases, an auxiliary stationary motor, set at the top of the hill, which could be operated from the same circuit as the locomotive, would overcome these difficulties.

The motor shown in the illustration was recently made for a Western coal-mine for this purpose. The outside dimensions of the hoist are 9 feet by 7 feet 4 inches, and the total height is nearly 6 feet. The motive-power is furnished by a Thomson-Houston motor of 35 horse-power, and wound for a potential of 440 volts. It is series-wound, and runs normally under load at a speed of 1,100 revolutions per minute. The speed from the motor-armature is reduced so as to give a rope-speed of

a little over 500 feet per minute. The drum, which is mounted on a sleeve and is thrown into action with the large gear by a friction-clutch, is 4 feet in diameter, with a 36-inches face. It is equipped with five controlling-levers, one of which operates the rheostat contact, one the drum clutch, two are brake levers, which are entirely independent one of the other, and the last throws the reversing-switch, giving the drum, when thrown into action with the motor, positive motion in either direction of rotation. J. W.

ACCIDENTS DUE TO FIRE-DAMP IN PRUSSIAN MINES DURING 1891.

Die im Jahre 1891 *auf den Steinkohlen-Bergwerken Preussens durch schlagende Wetter verursachten Unglücksfälle.* ANON. *Zeitschrift für das Berg-Hütten-und Salinen-Wesen im preussischen Staate, vol. xl., pages* 41-58.

During 1891, 104 fire-damp explosions—101 resulting in more or less serious injury to life and limb, 1 having no such results—occurred in 65 various collieries in the kingdom of Prussia. Of these explosions, 85 took place in the Lower Rhine-Ruhr coal-field, 5 in the Aix-la-Chapelle coal-field, 3 in that of Saarbrück, 7 in Lower Silesia, and 1 in the North German Wealden coal-mines. As mentioned elsewhere, 132 persons lost their lives thereby, 46 others were seriously injured, and 138 slightly hurt. It is noticeable that six fatal explosions (an average of one apiece) occurred in six mines of Lower Rhine-Ruhr, where the coal worked is apt to make much dust.

The immediate causes of explosion are classified as follows :—

	Cases.
Use of naked lights	11
Use of matches, tobacco pipes, etc.	5
Unauthorized opening of safety-lamps	7
Damaged condition of safety-lamps, or injury to same in course of use...	12
Sparks from soot or oil clinging to the gauze of the safety-lamp	1
Flame blown through gauze of safety-lamp by too great rapidity of air-current	5
Flame striking through gauze, owing to careless handling of safety-lamp	25
Shot-firing...	36

As to coal-dust, the state of the workings at the time of explosion is described as being damp in 24 cases, dry and not especially dusty in 58 cases, dry with more or less dust in 19 cases, and 1 case unknown. The worst explosion, causing 57 deaths, took place in a dusty mine. In 10 fatal explosions, one result of the accident was to coke the dust present in the mine.

As to atmospheric conditions, sudden changes took place in the barometer-level at the time of 7 explosions, change in air-temperature only in 1 case, and change both in air-temperature and barometer-level in 2 cases.

The greatest number of explosions occurred in March, and January was, as in previous years, an exceptionally unlucky month.

The origin of the fire-damp is ascribed to slow, continual streaming in of the gas in 58 cases, to secondary accumulations in 5 cases, to sudden fall of rock in 1 case, to blowers in 25 cases, to sudden issue from fissures in 10 cases, and from a borehole in 1 case ; the origin is unaccounted for in 2 cases.

To the above numbers must be added 4 cases of suffocation of miners by fire-damp in the Lower Rhine-Ruhr coal-field without the occurrence of an explosion. O. S. E.

THE FIRE IN THE FANNY-CHASSÉE COAL-PIT, LAURAHÜTTE, UPPER SILESIA.

Der Grubenbrand auf der Steinkohlengrube Fanny-Chassée bei Laurahütte O.-S. By— FIEBIG. *Zeitschr. für das Berg-Hütten-und Salinen-Wesen im Preuss. Staate*, 1892, *vol. xxxviii., pages* 291-295.

In the Fanny-Chassée coal-pit three seams are worked, being in ascending order :—

The Caroline seam	19½ to 24 feet thick.
The Glück seam	6½ feet thick, and
The Fanny seam	26 feet thick.

Between the two uppermost seams the barren rock is made up of strongly bituminous shales, with much pyrites, 13 feet in thickness ; while between the Glück and the Caroline seams occur shales and square-jointed sandstones 82 feet thick. In all three coal-seams numerous fissures traverse the mass, filled with fine coal-dust.

The sinking of the shaft in the first years of this century was attended with considerable difficulties, owing to the many water-feeders and the unfavourable lie of the strata. These difficulties were presently complicated by the fire, which suddenly blazed up with great fierceness in the workings. At one time the fire was attributed to the masses of burning cinders from a waste-heap (of the neighbouring Glauben zincworks) which fell in with a portion of the measures on the breaking down of the pillars near the outcrop of the Fanny seam. Dams of timber and masonry plastered with lime proved ineffectual to arrest the fire, which forced its way through the fissures in the coal. Large areas were successively given up to the flames. Then spherical iron dams were used, and water conveyed from the surface by the help of steam-power was poured on to the workings behind the dams : but the pressure of the vast quantity of water thus ponded back broke through the dams and pillars, and the water itself, taking up the sulphureous products of combustion, became sufficiently acid to corrode the pumping gear.

Proceeding still on the assumption that the fire had been communicated by some means from aboveground to the goaf, the engineers left in all three seams a pillar of coal 11 feet thick standing right round the burning area, and in all headings meeting this pillar masonry-dams were built with great care. It now seemed that the fire had been effectually cut off, and in some places the coal was worked right up to the boundary pillar without any further interruption.

Suddenly, however, the flames burst forth in the Fanny seam, in a portion of the workings which had but lately been worked, and was entirely shut off from the old area of conflagration. It now became evident that the origin of the fire was not external, but was traceable to spontaneous combustion of the coal. Large masses of coal brought to day were, as an experiment, thrown together in great heaps ; contrary to expectation, however, they did not burst into flame. Then it was pointed out that the fire had on no occasion originated in the Caroline seam, and experiments soon proved that the decomposition of the pyrites contained in the bituminous shales lying betwixt the two other seams was the true cause of the fire. This fact being placed beyond doubt, the methods of working and ventilation were altered ; it was important that good use should be made of the sort of truce which intervenes between the opening up of a new district and the actual outbreak of fire. The workings are divided into smaller areas than formerly, and at certain points dams are built in the self-acting planes and in the bord-ways, leaving just the necessary openings for haulage and for passage of air-currents ; bricks and mortar lie ready

to hand for stopping up these openings at short notice. The coal being so largely fissured, roof, floor, and working-face are plastered with mortar for considerable distances. Where small ventilating shafts are not found sufficient to secure a regular air-current, Körting's steam spray-injectors are used. If fire breaks out, it is soon circumscribed by the airtight dams, and dies down of itself for want of oxygen. When the thermometer on the hither side of the dam shows a sufficient fall of temperature, an opening is made, the noxious gases driven off by fresh air-currents, and the district examined with safety-lamps. All being well, the workings are carried on and continued till the next outbreak of fire. O. S. E.

ON THE ANALOGY IN MODE OF OCCURRENCE BETWEEN THE NATURAL GAS OF THE UNITED STATES AND FIRE-DAMP OF BELGIUM.

Sur les Analogies de Gisement du Gaz Naturel aux Etats-Unis et du Grisou en Belgique. By — LOHEST. *Annales de la Société Géologique de Belgique, vol. xix., pages* 44-47.

In the United States, gas and petroleum occur in anticlinal folds; in the Belgian coal-fields, plicated strata contain the most fire-damp. In the Mons basin, for example, the southern portion, where the structure is very complex, is subject to instantaneous disengagements of fire-damp. Towards the north of the basin, however, where the beds lie flat, there is little fire-damp. Similarly, in the Liége basin, fire-damp is most abundant where the strata are most disturbed. The roofs of the coal-seams are compared with the impermeable shales which cover the porous gas-yielding limestones, and the coal itself, fractured and full of joints, to the latter.

While the author thus thinks it possible that the fire-damp may be concentrated in the anticlinal folds, as in America, he admits the difficulty that the fractures of such folded beds would tend to facilitate the escape of the gas, while in the horizontal strata it might rather be expected to be retained.* G. W. B.

A FIRE-DAMP INDICATOR.

Note sur un Nouvel Indicateur de Grisou. By G. CHESNEAU. *Annales des Mines, series* 9, *vol. ii., pages* 203-221, *and* 2 *plates.*

Mr. Chesneau, after describing the dangers and inconveniences connected with the use of the Pieler alcohol lamp and of different improved models of that lamp, describes an alcohol lamp of his own invention.

It is composed of a brass reservoir for the alcohol, surmounted by a circular crown for the admission of air (which can be regulated) through double gauzes. Resting on the crown and surrounding the wick-tube is a solid cylinder of sheet-iron which serves as a screen. Above this screen, and resting on it, is an iron-wire gauze 5½ inches high. The gauze is surrounded by a sheet-iron shield furnished with an observing-window, completely closed by a sheet of mica, on which a scale is engraved. The shield is fitted at its base with an annular diaphragm which closely surrounds the base of the gauze so that the exterior air can never arrive directly to the gauze. The diaphragm rests on an asbestos washer for the purpose of diminishing the heating of the lower part of the lamp when it is in a gaseous mixture and the shield gets hot. The top of the shield is furnished with openings protected by

*[But if fire-damp be a consequence of partial distillation of coal its presence in disturbed strata is easily accounted for, as being the result of the heat produced by thrusting and folding.—ED.]

screens to prevent currents striking the gauze. An outer movable screen of sheet-iron having a window, protects, when required, the base of the observation-window in the shield against currents, and this prevents the deposit of dew which tends to form on the interior of the mica by outside cooling.

To make an observation, it suffices to bring the window in the movable outer shield in front of the mica observation-window. In a calm atmosphere the dew disappears of itself some minutes after lighting, and only re-forms when the lamp is brought into a cold current.

The interior of the reservoir contains a small piece of cottonwool under the wick-tube to prevent the rapid escape of alcohol if the lamp is overturned. When the reservoir of the ordinary Pieler lamp is tightly packed with cottonwool it only gives a cap in gaseous mixtures for some minutes, because the alcohol retained by the cotton ascends the wick with difficulty, and the flame soon lowers, and the wick carbonizes. The almost total suppression of the cottonwool in the new lamp offers no danger, because the lamp goes out, when laid horizontally, before the alcohol can spread in the lamp. It is not so with the ordinary Pieler lamp, where the alcohol can spread and burn on the gauze when they are filled and much inclined. The alcohol is introduced by a hole, fitted by a screw, a tight joint being obtained by means of a washer of lead or leather. The weight of the lamp, full of alcohol, is 3·2 lbs.

The entry of the air and the exit of the products of combustion being absolutely separate in the new indicator there can be no mixture between them as in the Pieler lamp, in which air charged with gas may come in and burn in the whole height of the gauze and heat it strongly. Placed in stationary gaseous mixtures of increasing percentages, the lamp gives caps which only reach the top of the gauze with about 3 per cent. with the alcohol flame regulation adopted. With from 3 to 5·5 per cent. the alcohol flame elongates, the cap enlarges, becomes cylindrical, but lowers more and more from the summit, the lamp being unable to emit the increasing products of combustion. A little above 5·75 per cent., the alcohol flame elongates towards the top of the gauze, but without sensibly heating it, the quantity of air drawn in being insufficient for the complete combustion of the gas, and the alcoholic vapour liberated by the heating of the lamp. Above 5·75 per cent. (proportion inferior to the limit of inflammability of air and fire-damp, which is produced with 6·1 per cent.) all flame disappears in the lamp, and the gas burns with a very paleflame in the crown only without considerably heating the reservoir. All is extinguished after some seconds very rapidly if the inlet regulator is closed. The alcohol flame of the lamp reduces rapidly to its initial height when removed to pure air. Tested in explosive currents the lamp appeared as safe as ordinary safety-lamps of good construction.

The height of the caps vary considerably according to the volatility of the alcohol employed, and to obtain comparable results the same alcohol must be used. The alcohol giving the best results is wood spirit or methylated alcohol of 92·5 degs. by the Gay-Lussac alcohol-meter at 59 degs. Fahr. To increase the visibility of the caps, chloride of copper may be dissolved in the alcohol with a little hydrochloric acid to maintain it in solution. The proportion recommended is about 17 drops of a saturated solution of crystallized chloride of copper in concentrated hydrochloric acid per pint of alcohol; this gives the alcohol flame a green tinge. In this way caps may be seen with from 0·1 to 0·2 per cent of fire-damp. They are easily seen with 0·5 per cent. or more,

The proper methods of regulating the alcohol flame and the appearance of the caps are minutely described. W. N. A.

CLASSIFICATION OF ORE-DEPOSITS.

By J. F. Kemp. *School of Mines Quarterly* (*New York*), *vol. xiv., pages* 8-24.

This paper gives a *résumé* of the various attempts at the classification of ore-deposits which have from time to time been made, from von Cotta downwards. The author decides in favour of schemes based entirely on origin, as opposed to those which depend more or less completely on form or character of filling; and proposes a detailed classification which divides ore-deposits into four classes, viz. :— (*a*) Of igneous origin ; (*b*) deposited from solution ; (*c*) deposited from suspension ; (*d*) of indeterminate origin. These classes, especially (*b*), are of course largely subdivided. G. E. C.

THE ORIGIN OF CALAMINE-DEPOSITS.

Note sur l'Origine des Gîtes Calaminaires. By A. Lodin. *Bulletin de la Société Géologique de France, series* 3, *vol. xiv., pages* 783-793.

The author first points out that two theories have been successively proposed to account for the origin of the deposits of carbonate, hydrated carbonate, and anhydrous (or, as the case may be, hydrated) silicate of zinc, collectively known to miners as calamine. The oldest theory postulates the derivation of these ores from pre-existing sulphides, altered by atmospheric agency. This hypothesis is based on the almost constant occurrence, in the deposits, of blende, galena, and pyrites, in a quantity which increases directly as the depth from the surface.

The other theory, originated by Elie de Beaumont and developed by Delanoue, postulates the uprise of acid waters containing zinc in solution, which is deposited on contact with limestone. In that case, the presence of blende and the accompanying sulphides would be due to the reduction and sulphurization of oxydic ores by organic matter. One of the chief arguments adduced in support of this theory was the presumed refractoriness of blende to atmospheric agencies, but facts do not accord with this presumption : thus, at the Pontpéan mine, some heaps of blende which have been exposed to the air for about a century have finally lost all their zinc, leaving only a plumbiferous and argentiferous residue.

The author's study of several calamine-deposits, combined with a series of experiments, leads him to reject absolutely the last-mentioned theory and to confirm the older hypothesis. He draws attention to the occurrence of calamine as being exclusively in calcareous rocks, never in siliceous or alumino-siliceous rocks, and concludes that the deposition of the ore is due to chemical precipitation by calcium carbonate. He describes in detail the principal features of a calamine-deposit, taking as examples those of Sardinia, and points out that their constitution is closely connected with the actual surface-features of the district and with the mean level of the underground waters (*la nappe souterraine*).

The reaction of water, holding oxygen in solution, on blende-galena deposits produces (*a*) ferric hyposulphate and lead sulphate, (*b*) zinc sulphate and sulphuric acid. Group *a* remains *in situ*, and is gradually decomposed by the action of oxygenated and calcareous waters into iron oxide and lead carbonate ; while group *b* tends to percolate downward through the rocks.

Now zinc sulphate in concentrated solution is precipitated by calcium carbonate, the results of the reaction being a mixture of hydrated calcium sulphate and zinc hydrocarbonate (hydrozincite). The experimental difficulties of producing also the anhydrous carbonate (Smithsonite), which occurs in conjunction with hydrozincite in calamine-deposits, appear to be insuperable so far, but its occurrence in nature may be explained as follows :—

1.—Presence of a large excess of sulphuric acid or of very acid ferric sulphate; consequent formation of gypsum and absorption of sufficient water to prevent the formation of hydrocarbonate.

2.—Presence of carbonic acid gas at a sufficient tension to prevent the hydration of the anhydrous carbonate by water in excess. (At the calamine mines of Malines, in the Department of the Gard, carbon dioxide often comes off in such quantities in the neighbourhood of great masses of calamine and blende that the work of quarrying through the limestone to get at the ore has to be suspended for a few days).

3.—Dissolution of the hydrocarbonate in water holding carbonic acid in excess, then precipitation (from this solution) of the anhydrous carbonate by excess of calcium carbonate. To this reaction the author would largely attribute the origin of the veins of pure Smithsonite which are often found in the hanging and foot-walls of the deposits.

As to magnesian limestones, the author found that giobertite (natural carbonate of magnesia) gave no precipitate with a sulphate of zinc solution; but powdered dolomite, after a prolonged contact, gave a very feeble precipitate.

The origin of the silicates in calamine-deposits is no doubt due to the presence of hydrated or combined silica in the limestones, which are acted upon by the sulphate of zinc; and, given that condition, the author considers the formation of zinc silicate just as easy of explanation as the formation of zinc hydrocarbonate.

O. S. E.

THE PERSISTENCE OF ORES IN LODES IN DEPTH.

By Wm. P. Blake. *Engineering and Mining Journal* (*New York*), *vol. lv., page* 3.

Prof. Blake discusses the question of the relative mineralization of lodes at great and shallow depths, in the light of increasing evidence that it has been effected, in the majority of cases, by lateral secretion. The presumption is that the solvent action and movement of the solutions would be greatest comparatively near the surface, yet this does not necessarily affect the mineralization, as cases are cited of lodes which in depth have changed from wet to dry, without any falling off in the productiveness of the lodes. In the Butte district there are sometimes evidences of active oxidation far below the water-level; and there is no ultimate tendency to decreasing richness in depth.

Apart from the outcrops of lodes, which are affected by special climatic and other agencies referred to in detail, the conclusion is that, in a homogeneous country-rock, the productiveness of lodes may be expected to continue to as great a depth as is likely to be reached in practice.

G. E. C.

FLORA AND FAUNA OF THE BELGIAN COAL-MEASURES.

Matériaux pour la flore et la faune du houiller de Belgique. Par X. Stainier. *Annales de la Société Géologique de Belgique, vol xix., pages* 333-359.

The author points out the importance of a knowledge of the flora and fauna in the study of the origin of coal, and in the practical question of the correlation of seams.

An important result of the investigation is the establishment of the fact of the abundance of fish-remains in the Belgian coal-field, as hitherto they have been rarely recorded. The roofs of many of the coal seams contain numerous scales and other remains. One is described as a veritable bone-bed. It occurs over the first

workable seam in the Upper Coal-measures, and the author suggests its correlation with the bone-bed in Yorkshire.

A table of fossils, classified according to their horizons in the Coal-measures, is given.

In the Sainte-Barbe seam, calcareous nodules containing *Goniatites* (*diadema*, and *Listeri*) are found. Some of these Goniatites contain carbonaceous matter. The presence of marine—or even pelagic—organisms, such as Goniatites, in the very midst of a coal-seam, is important in connexion with theories of the origin of coal.*

G. W. B.

MINERAL DEPOSITS OF THE CAUCASUS.

Note sur les Principaux Gisements Minéraux de la Région du Caucase. By — Leproux. *Annales des Mines, series* 9, *vol. ii., pages* 491-540, *and plate XIII.*

The mountain-chain of the Caucasus, formed near the end of the Tertiary period, runs about west-north-west to east-south-east. Between it and the Anti-Caucasus to the south there are two main valleys sloping to the Caspian and Black Sea respectively. In these a series of Tertiary formations are developed, covered towards the Caspian by loess. The Tertiary rocks are succeeded by some little-developed cretaceous beds; next come Jurassic rocks, often traversed by sheets of diabase and melaphyre; and then a considerable thickness of Palæozoic clay-slates of doubtful age, which rise to the north and south against the crystalline rocks forming the axis of the chain.

Iron.—This is fairly abundant, but too far from the coast for export; and the extensive destruction of the forests, and the scarcity and poor quality of the coal, render smelting impossible, unless petroleum can be utilized.

Manganese.—The deposits are of considerable importance, especially those on the river Kvirilla in the district of Charopan, where the mineral-bearing country comprises an area of 130 square kilometres. The beds, accompanied by limestone and sandstone of the Upper Eocene, rest on the Upper Chalk. All have remained horizontal, and are well exposed in section by the deep valleys of the Kvirilla and its affluents.

The following analyses, among others, are given:—

Manganese	54·86	...	59·05
Oxygen (combined with manganese) ...	31·84	...	24·25
Iron	1·50	...	1·00
Waste	11·80	...	5·70

The average Kvirilla ores give 55 per cent. of manganese, 0·15 to 0·20 per cent. of phosphorus, and 2 to 6 per cent. of silica.

The production has been approximately as follows:—

	Tons.			Tons.
1884 . .	17,000	... 1887	...	53,000
1885 ...	56,000	... 1888	...	30,000
1886 ...	68,000	... 1889	...	70,000

The cost of raising is stated as 7s. 6d. per ton, and transport from the mine to Kvirilla station 21s. per ton.

Cobalt.—Several deposits of cobaltine exist in the government of Yélizavetpol, of which the most important is that of Daschkessan. This occurs under a sheet of diabase carying much oxide of iron; the cobaltine is contained in the rock forming the wall, together with garnets and copper pyrites.

[* Marine organisms may be washed up on to a beach or into an estuarine sandbank or marsh after great storms in fair abundance. Nodules prove nothing as to where the organisms lived.—Ed.]

Lead and Silver.—Many lodes carrying argentiferous galena traverse the crystalline rocks forming the axis of the chain and the Palæozoic rocks immediately above them, but the only mines of any importance are those of Sadone, near Vladikavkaz. These produced, in 1888, 5,660 tons of mineral containing 0·02 per cent. of silver, 6 to 7 per cent. of lead, and 0·06 per cent. of copper. They belong to the Government, and the ores are treated near by.

Zinc.—Only one mine is being worked, that of Khot-Eli, south of Batoum, the ore from which runs 12 per cent. of zinc and 6 per cent. of copper, but only 125 tons were exported in 1888.

Copper.—A great many deposits of copper are known throughout the Caucasus, the most important of which are those of Alverdski, Kedabek, and Zangezour, on the north side of the Anti-Caucasus, in the Governments of Tiflis and Yélizavetpol. These deposits are rich, but situated in a very difficult and elevated region, where the only fuel obtainable is petroleum residue, which at Kedabek costs considerably over £3 per ton.

The deposit of Alverdski appears to be a large irregular mass at the contact between diabase and quartzite. The ore consists of copper pyrites, grey copper and erubescite, in a gangue of iron pyrites and gypsum, and varies in percentage from 8 to 25, with an average of 10 per cent.

The deposits at Kedabek are also irregular masses, the nature of which is not definitely indicated. They are composed of a quartzose rock sprinkled with pyrites and black oxide of copper.

The following analyses, among others, are given :—

	Upper Mass. Large Grains.	Karlstock Average.	Arnold-stock. Rich Ore.	50 yards, south of Upper Mass. Average.
Silver	0·02	—	—	—
Copper	13·32	9·29	22·05	7·12
Iron	34·38	15·66	14·31	25·92
Zinc	—	1·87	5·82	5·10
Sulphur	38·85	22·32	23·74	25·80
Insoluble matter	11·29	46·46	30·38	15·25
Manganese, antimony, etc.	3·12	—	—	—

The deposits of Zangezour, near the Persian frontier, consist of a breccia about 2 feet thick, at the contact between granitic syenite and diorite, both of which are mineralized. It contains granite cemented by a ferruginous clay, both carrying malachite, copper oxide, native copper, etc., the average copper contents of the whole being about 10 per cent.

The total production of copper was 1,500 tons in 1889. In 1838, it was 24,000 tons of ore.

Gold has been found *in situ* in a few places, and in the gravels of some of the rivers, but never in workable quantity.

Petroleum.—Deposits exist all round the Caucasus chain where it merges into the low country, but are of most importance at the two extremities, the peninsulas of Tamansk and Apscheron, especially the latter, where the Baku wells are world-famous. The petroleum is generally, though not exclusively, found in the Upper Tertiary series.

Contrary to the generality of Caucasian petroleum, that of the Tamansk peninsula is light (density 0·755 to 0·825). It is contained in thick beds of clayey sand of Tertiary age, and is very irregular in distribution. In the region to the south, the petroleum is heavy and thick (density 0·945 to 0·985).

The Baku deposits are practically confined to the Apscheron peninsula, the formation of which is entirely Tertiary and post-Tertiary; and are generally found in the southern and eastern portions of the anticlinal folds. The petroleum is usually oily and thick, of brown or black colour, but there are great differences in the density. In some cases it is accompanied by salt water, the absence in which of bromine and iodine indicates that it does not come from the Caspian, as had been supposed. The total production for the region was 3,316,004 tons in 1889, of which 3,282,633 tons was from Baku.

Coal.—Coal-beds are very numerous, being found almost wherever the Jurassic formation crops out, but they are thin and of poor quality. A number of localities are described.

The total production in 1888 was 8,522 tons, including lignite, etc. G. E. C.

THE MINERAL POVERTY OF WESTERN BORNEO.

Notes Recueillies au Cours d'une Exploration dans l'Île de Bornéo. By — CHAPER. *Bulletin de la Société Géologique de France, series* 3, *vol. xix., pages* 877-882.

The author went up the great river Kapoeas on a trading steamer from Pontianak to Sintang (240 miles), and then in boats manned by natives to Semitau (72 miles). The banks are low and apparently made up of yellowish clay; indeed, the whole plain-country is covered by a deposit of fine clay (about 14 feet thick), and this extends up the hillsides to the 200 feet level. Below the clay are yellowish sandstones and soft shales, with which are intercalated two or three bands of hard conglomerate. Near Mounts Oejan and Rajoen, the rocks are intensely metamorphosed into quartzite, jasper, and dark green diorite with garnets. In both the Kapoeas and Sebroeang valleys there are below the clay deposits of very fine white auriferous and diamantiferous sand. Some of these deposits are still washed for gold by natives and Chinamen, but the profits are comparatively small. As to diamonds, the author purchased three at a high price at Samarangaï, but there is strong reason to believe that they had been brought to Borneo from some other country. Lignites of fairly good quality occur in plastic clay in the Knapei Valley and at Selimbau, in localities which are under water in the rainy season. At Blietang there are several bands of lignite, one of which is said to be 15 or 16 feet thick; but here also, the seams are under water when the river is high.

To sum up, that part of Western Borneo which came under the author's observation appears to offer little scope for industrial enterprise; and the chief hopes of explorers must be directed to the still unknown, mountainous centre of the island.

O. S. E.

ECONOMIC RESOURCES OF SIKKIM.

Notes on the Geology and Mineral Resources of Sikkim. By P. N. BOSE. *Records of the Geological Survey of India, vol. xxiv., pages* 217-230, and map.

The mineral wealth of the district consists chiefly of copper ores. They occur in a series of rocks known as the Daling Group, and are disseminated through the schists and slates in bands and veins parallel to the bedding. In one case a true lode occurs. The ore is copper pyrites, often accompanied by mundic. Traces of sulphate, carbonate, and oxide of copper frequently occur as the result of the alteration of the pyrites. The gangue is shale hardened by the infiltration of quartz, or of quartz alone. In the junction-beds and in the underlying gneiss

itself, copper is also found, but the ores are less rich. In the Daling Group the richest ores occur in greenish-grey, rather soft slaty shales. The ores are worked by the natives, but, owing to their primitive methods, they are compelled to abandon their mines as soon as they get to a certain depth, on account of the volume of water. With improved appliances, the deeper, and apparently richer, deposits could be reached.

Analysis of a sample of ore taken at random from one of the workings gave 20·31 per cent. of copper. An average sample from another working gave 12·21 per cent. Picked ore from the Ráthokháne locality was found by Mr. Mallet to contain 8 or 9 per cent. G. W. B.

SWAMP OAK AND NIANGALA GOLD-FIELDS, NEW SOUTH WALES.

Geological Notes on the Swamp Oak and Niangala Gold-fields. By G. A. Stonier. *Records of the Geological Survey of New South Wales, vol. iii., pages* 60-68.

The Swamp Oak gold-field is situated about 25 miles east of Tamworth, New South Wales, and that of Niangala 9 miles farther south-east, a little west of the watershed of the Great Dividing Range.

Alluvial gold has been found since 1852 in Swamp Oak Creek, and there is auriferous Tertiary alluvium, overlaid by basalt, at Niangala, which, however, has been little tested as yet.

The gold-bearing reefs are contained in Lower Carboniferous mudstones; the whole series is seamed with quartz, but the veins only carry gold in the vicinity of diorite-dykes, which occur in a well-marked axis of upheaval which has probably occasioned the veins.

The auriferous belt at Swamp Oak is $2\frac{1}{4}$ miles long and $1\frac{1}{2}$ miles wide. The reefs cannot be traced any distance—this being apparently due to closing of the walls and not to complicated faulting. They mostly dip towards the dyke. They all carry mispickel in isolated crystals, averaging $1\frac{1}{2}$ per cent. of the bulk. Iron pyrites is exceptional. The gold is fine, and mostly in combination with the mispickel. So far as the district has been opened out the gold seems to run in shoots, not, as a rule, exceeding 20 feet in width. The average thickness is 9 inches to 1 foot, the veins themselves running up to 3 feet. The distance between the shoots and their dip vary considerably.

The Niangala district appears to resemble that of Swamp Oak in general character. G. E. C.

GRAPHITE FROM CEYLON.

On Veins of Graphite in Decomposed Gneiss (Laterite) in Ceylon. By Dr. Johannes Walther, *translated by* R. B. Foote. *Records of the Geological Survey of India, vol. xxiv., pages* 42-45.

The most important mineral export from Ceylon is graphite. In the virgin forests mighty domes of grey gneiss weathered to a depth of 39 feet, and soft enough to be cut with a knife, occur. In this decomposed gneiss, or laterite, many branched veins of graphite are found. The main veins are from $4\frac{1}{2}$ to $8\frac{1}{2}$ inches in thickness. As to the origin of the graphite, it is supposed that the fissures originated in the unweathered gneiss by dislocations, and that they were afterwards filled by sublimation of some hydrocarbon rich in carbon. G. W. B.

THE OFOTEN RAILWAY, AND THE IRON-ORE DEPOSITS OF SWEDISH LAPLAND.

Ofotenbanen og Jernmalmfelterne i Svensk Lapmarken. *By* Prof. AMUND HELLAND. *Norsk Teknisk Tidsskrift, vol. x., pages* 1 *to* 13, *and two plates.*

As is well known, all work on the Ofoten-Luleä railway, projected to open up these deposits, is now stopped, and the English company which held the concession has forfeited it. The section from Gellivare to Luleä, on which a large amount of work has been done, will probably be taken over and finished by the Swedish Government. The parliament has already granted seven million kroner (nearly £400,000) for this purpose.

The known iron-ore deposits of importance are four in number—Gellivare, Kirunavara, Luossavara, and Svappavara; of which the latter is of little immediate interest, as it is far from the projected line of railway. The standard description of these deposits is that of the special commission sent by the Swedish Government to examine them in 1875, and published by the Swedish Geological Survey in 1877.

Gellivare is in the extreme north of Sweden, 207 kilometres north-north-west of Luleä, on the Gulf of Bothnia. The ore-deposits occur in an irregular line of low hills, 4 or 5 kilometres in length, rising from the surrounding marshy land and forest, the ore being traced by its actual outcrop where the rock is bare, and by magnetic survey where the surface is covered with soil and gravel. The country-rock is gneiss, in which ore-layers of various size occur, and which includes dykes and masses of granite. The ore and gneiss are interbedded, and in places are curved and contorted together; the strike varies, but generally approaches north-east, with a south-easterly dip. The ore is principally magnetite, with a varying admixture of red hæmatite.

Of 41 general average samples taken by the Commission, 26 had over 70 per cent. of metallic iron; 13 gave from 60 to 70 per cent, and only 2 gave under 60 per cent. It is free from visible pyrites—9 only of the samples giving over 0·05 per cent. of sulphur, while 12 had none at all; but in places it shows considerable amounts of apatite sprinkled throughout the mass. Of the 41 samples taken, 10 showed less than 0·05 per cent. of phosphorus, 3 between 0·05 per cent. and 0·1 per cent., and 21 over 0·1 per cent., the highest being 0·7 per cent.; and it appears that the ore from the same mass contains largely varying amounts of phosphorus, which is confirmed by the experience of the charcoal-iron smelters, who have taken ore from here on a very small scale for over 100 years.

It will be seen that the Gellivare deposit is not, as before reported, a whole mountain of iron ore; it is only a mountain containing ore-belts and masses; but these are of great size, up to and over 100 feet in width, and include 5 per cent. of the surface over which they occur. The total area of the ore-masses is reckoned at 652,310 square metres (over 160 acres), or more than that of all the other deposits of Lapland put together; of this, 94,673 square metres is considered to be free from harmful impurities, and in belts of sufficient width. At 5 tons to the cubic metre, this should produce 473,365 tons of good ore for each metre sunk, or nearly seven times as much if the inferior ore is included.

The Kirunavara deposit is 312 kilometres from Luleä, in the same north-north-westerly direction, and is only 182 kilometres from the western coast of Norway, at Ofoten. There is here a hill over 3 kilometres long, running almost north and south, the mid ridge of which consists of solid iron ore, from 55 to over 230 metres wide, standing out in bold crests.

This deposit was known as early as 1736. The rock is greyish or reddish porphyritic *hälleflinta*, and the ore-mass, consisting exclusively of magnetite, is nearly vertical. Of 28 general samples taken by the Commission, 12 contained from 70 to 73½ per cent. of iron; 13 from 65 to 70 per cent.; and only 3 were poorer; the lowest result being 61½ per cent. The amount of sulphur is small, varying from 0·03 to 0·15 per cent., but the phosphorus is in places considerable. Only 4 samples gave from 0·03 to 0·047 per cent., the other 24 ranging from 0·305 to 2·802 per cent., with an average of 1·396 per cent. It is remarkable that all the samples from the middle of the deposit gave the highest results for phosphorus, while those taken farther north or south, towards either end, were progressively poorer, until ore of excellent quality, practically free from phosphorus, is reached.

As this deposit crops out at the surface along its whole length, and is free from detrital covering, its extent cannot be doubted. Its area is calculated at 311,000 square metres, the length being 4,157 metres, and the average width about 75 metres; at 5 tons to the cubic metre this would produce 1,555,000 tons for each metre in depth removed; and it is estimated that the enormous amount of 260,675,000 tons exists in the hill above water-level, available by open quarrying.

The Luossavara deposit is only 2 kilometres north of that just described, on the other side of a small lake, and in almost the same line of strike. The deposit here is in a conical hill, much covered with detrital matter and trees; the ore only crops out at the surface as a large mass at the top of the hill, its extent having been defined by compass surveys. These show its length to be 1,336 metres, and the breadth at the top is 46 metres, the area being taken at 49,276 square metres. This would give 246, 380 tons of ore for each metre of deposit removed, and it is estimated that 27,679,700 tons exist in the hill above water-level.

Of 8 general samples, 5 gave from 70 to 73 per cent. of iron, and 3 between 67½ and 69 per cent; the sulphur varying from 0·03 to 0·09 per cent.. For phosphorus, 2 gave under 0·01 per cent., and 4 others under 0·03 per cent., the remaining 2 giving 0·057 and 0·082 per cent.

Svappavara is about 60 kilometres north-east of Gellivare. Its iron-ore deposits are extensive, being 623 metres long by 53 to 98 metres broad, or an area of 37,376 square metres, so that each metre in depth should yield over 180,000 tons of ore, and it is estimated that between 66 and 67 million tons are available above the water-level. The ore is magnetite with hæmatite, containing from 50 to 73 per cent. of iron; sulphur is absent, but the ore contains from 0·129 to 1·547 per cent. of phosphorus. The deposit, being far from the projected line of railway, is now of little interest.

Prof. Helland next discusses the present sources of iron ore in Europe, and concludes that they are rapidly becoming inadequate to meet the demand, and that the British ironmasters in particular are likely to need large amounts of rich ore from new sources. He considers that they can best be supplied from these deposits, and that an output of some millions of tons per annum could not be considered as over-production. He then discusses the present condition of the projected railway from Ofoten on the Atlantic side to Luleä on the Gulf of Bothnia, which would open up the deposits, and shows that much work has been done on it already, and that the section from Gellivare to Luleä will probably be completed by the Swedish Government. The paper, which is accompanied by two good maps, is to be continued with a consideration of the cost of extracting the ore and of shipping it to foreign ports. A. L. C.

ON THE FORMATION OF THE MOST IMPORTANT GROUPS OF IRON-ORE DEPOSITS IN SCANDINAVIA.

Om dannelsen af de vigtigste i Norge og Sverige representerede grupper af Jernmalmforekomster. *By* J. H. L. Vogt. *Geologiska Föreningens i Stockholm Förhandlingar, vol. xiv., pages* 211-248.

This number contains an exhaustive examination and classification of the iron-ore deposits of Norway and Sweden, from a theoretical standpoint. It ends with a complete summary of the whole paper in German. A. L. C.

THE IRON DEPOSITS OF EASTERN SPAIN.

Sobre los Criaderos de Hierro y de Plomo del Levante de España. *By* Juan Pié y Allué. *Revista Minera, vol. xliii., pages* 193-196 *and* 213-217.

The author deals only with the more commercially important iron deposits which are interstratified in the limestones, either contained entirely by them or at the contact with the slates. These deposits are of recent origin, contemporaneous with the present outline of the mountain-systems, and owe their origin to the thermal springs carrying iron in solution.

At the contact of the iron with the limestone or dolomite, and even separating the various kinds of ore in a deposit, it is common to find clayey bands analogous in chemical composition to the flucans of lodes; these doubtless result from the double decomposition in the limestone and dolomite and from the wear of the slates. The frequent presence of gypsum in the walls of the deposit is also easily explained by the action of sulphur-bearing waters on the limestones.

The following laws are found to obtain with reference to these deposits:—Their direction is always parallel to that of the range to which they belong; the dip is the same as the slope of the mountain; and the degree of inclination usually diminishes with increased distance from the summit.

Details of many of the more important outcrops in the Sierras Alhamilla, de Bédar, and de Enmedio, and the district of Morata are given in the paper, but are not here abstracted.

Without being equal to the best Bilbao ores, those of this region are suitable for all metallurgical operations. The sulphur and phosphorus are, with the exception of one district, below the limit allowed for first-class irons. The insoluble residue varies greatly, in some cases rendering careful washing necessary, whilst in others it is very low. The silica depends somewhat on the enclosing strata, the deposits enclosed in the Triassic ranges having less than those in the older rocks. The quantity of manganese is greater than at Bilbao.

The iron percentage, as would be supposed from the formation, is also variable. Where the metalliferous springs met with pre-existing cavities, the ore is of high quality; whereas where the limestone had to be dissolved away it falls into two classes of 47 per cent. and over 50 per cent. respectively. It is observed that where the deposits are largest the quality is best.

The essential difference between these deposits and those of Bilbao lies rather in their disposition than in their nature. Whereas at Bilbao we have a bed of great persistence, more or less horizontal, but of limited thickness, in Eastern Spain we have successive beds of less persistence, with approximately vertical inclination.

As to the quantities available, it is not possible to form any definite idea, but the region seems fitted to become the successor to Bilbao. The author mentions

estimates which have been made by various persons of the quantities available in different districts, amounting altogether to over 30,000,000 tons, and considers that these figures understate the case.

In subsequent numbers the geological conditions of the district are discussed at great length. G. E. C.

IRON-ORES OF SALEM, INDIA.

Preliminary Report on the Iron-Ores and Iron Industries of the Salem District. *By* THOMAS H. HOLLAND. *Records of the Geological Survey of India, vol. xxv., pages* 135-159.

Salem appears to be an important iron district. The chief ore is magnetite, which also contains the largest percentage of iron. A practically inexhaustible supply of ore, containing nearly 60 per cent. of metallic iron, is obtainable in the Salem district. The iron is very free from phosphorus and sulphur, although detailed analyses are wanting. Hæmatite is subordinate in importance. Ores formed by the weathering of hæmatite—limonite, brown hæmatite, clay-ironstone, etc.—occur, but, although used by the natives on account of their softness, they are of little importance. Chromite also occurs.

No manganese ores are known in the district. Corundum deposits, valuable possibly as a source of aluminium, occur.

These iron-ores are worked in a primitive and wasteful fashion by the natives, who make both wrought-iron and steel. Their methods are described as imperfect forms of the Catalan and bloomery processes of Europe and of America. The fuel used is charcoal, the harder and more slowly-growing native woods being preferred for this purpose.

Many thousand acres of available forest occur in the neighbourhood.

The following estimation of the timber supply per annum and the resulting yield of iron is given :—

	Timber. Tons.	Iron. Tons.
Salem talûq	5,736	400
Attûr „	833	58
Námakal „	5,366	416
Uttankarai „	8,140	625
Total	20,075	1,499

Investigation is required as to which of the native woods are the most suitable for the production of charcoal for iron smelting. G. W. B.

GOLD AND IRON-ORE IN ELDSLAND, TIERRA DEL FUEGO, ARGENTINE REPUBLIC.

Om recenta lager af jernmalm under bildning på Eldslandet. *By* ANT. SJÖGREN *and* C. JUL. CARLSSON. *Geologiska Föreningens i Stockholm Förhandlingar. vol. xiv., pages* 75-86, *and woodcuts.*

Mr. C. J. Carlsson was chief of an expedition sent in the latter half of 1886 to investigate part of Patagonia and Tierra del Fuego, which had been reported rich in gold. The expedition went from Punta Arenas, near the Magellanes Strait, by sea to Bahia Porvenir, on the north-west coast of Tierra del Fuego; thence it proceeded by Bahia Inutil to Bahia San Sebastian, on the east coast, from which, as a base, all the coast to lat. 53° 47′ south was investigated.

The rocks of the northern part of Tierra del Fuego are sandstones and conglomerates of Tertiary age, with occasional bands of lignite; and are thinly overlain by more recent (Quaternary) deposits. The Tertiary strata probably overlie schists of Cambrian age, as at Mendoza and other places in the Cordillera.

Gold is found in many places, on the sea-coast and in the beds of most of the rivers and streams coming down from the Cordillera, along the whole coast up to Santa Cruz in Patagonia, but seldom in paying quantities. The original source of the gold is as yet uncertain; but gold has been found in certain beds of the Tertiary strata, although in minute quantities only.

The gold is accompanied by magnetite and hæmatite, brownish-red garnet, dark hornblende, and titanite, forming a black ironsand, which is regarded as the best indication of gold. This ironsand is very common in the surface-deposits (Quaternary), especially on the seacoast and in some of the river-valleys, where it has been concentrated into considerable layers; as at Valle Cullan, where some Chilians have erected a dressing-works for its treatment; and at Nombre Head, where there is a layer about 1·20 metres thick, between the Tertiary and Quaternary deposits, only 4 or 5 metres from the surface. Similar sand is found along the whole line of the east coast, in a band from 0·01 metre to over 1 metre thick, and of varying width up to 10 or 15 metres, more or less mixed with quartz sand; and it is probable that similar beds exist at the sea-bottom for a considerable distance from the shore, as soundings often show black sand, and large amounts of it are cast up on the beach after strong easterly winds.

Gold sometimes occurs mixed with the sand; but it is more generally concentrated beneath it on the bed-rock, or in a cement of clay and small stones. Its appearance and composition (roughly, an alloy of 80 per cent. of gold, and 20 per cent. of silver) are the same over the whole district, except that in the interior the gold is generally coarser, nuggets weighing as much as 30 or 40 grammes having been found, while on the seacoast the largest hitherto found has not weighed quite 2 grammes. The deposits are extremely variable in value: one small layer gave 800 grammes of gold to the cubic metre of sand washed, but an average richness is about 15 grammes per cubic metre.

Mr. A. Sjögren suggests that the interbedded deposits of magnetite and hæmatite, so common in Sweden and North America, may have had an origin analagous to that of the beds of ironsand above described.

He thinks that an examination of the magnetite deposits, especially in their deeper workings, clearly shows them to have been formed contemporaneously with the enclosing quartzite, gneiss, or other strata, and that all the conditions would be met by supposing them to have been formed from the ferruginous detritus of older rocks, mechanically concentrated by running streams or by the waves of the sea.

A. L. C.

IRON-ORES OF MEXICO.

The Occurrence of Hæmatite and Martite Iron-Ores in Mexico. *By* Robert T. Hill. *American Journal of Science.* *Series iii., vol. xlv., pages* 111-119.

The paper refers to the descriptions by Mr. Birkinbine and Prof. Silliman of the famous Cerro de Mercado near Durango, and describes a similar deposit near Monclova, Coahnila, on the Mexican International Railway.

The mountain in which the deposit occurs is composed of nearly vertical strata of hard limestone of Lower Cretaceous age, traversed parallel to the stratification by vast masses of eruptive diorite, certainly of later origin.

The extensive talus of limestone *débris* around the base of the mountain is largely minged with rounded lumps of specular hæmatite. About 1,000 feet above the railway are exposed large masses of ore, in a line corresponding with the strike of the limestone-diorite contacts; resembling fragments of a vertical vein, but apparently non-continuous. Some are entirely in limestone, others entirely in

diorite, while several present a wall of each. They seem to have been formed by replacement of the limestone at the contact, or of the masses of limestone included in the diorite, and are certainly of later age than the enclosing rocks.

The ore has the external appearance of black magnetite, but the interior consists of bright specular hæmatite, studded with minute granules and octagonal crystals of martite.

In several outcrops about a foot of limonite is seen against the hanging-wall of limestone.

Many other similar deposits in Mexico are referred to, which in most cases seem to occur under the same geological conditions—folds of Cretaceous limestone, broken by diorite protrusions, and separated by valleys largely covered by the *débris* from the mountains.

The author is of opinion that these remarkable masses of hæmatite are likely to become an important factor in the future iron supply of the world.

G. E. C.

SAFETY NETS IN WINDING SHAFTS.

Sur le Filet de Sureté établi au puits Jules Chagot des Mines de Blanzy. **By** — LEBRETON. *Annales des Mines*, 1889, *series* 8, *vol. xv., pp.* 409-416, *and four figures.*

It sometimes happens that workmen fall into the shaft, either inadvertently or by a mistake in moving the cage. Sometimes the victim is killed, and at other times the injuries are slight, and it is possible to save him; if they are serious he may be drowned before it is possible to render him assistance.

At the Jules Chagot pit, a wire rope net has been placed about 2 yards below the keps at the lowest hanging-on, which is about 350 yards in depth.

The net, which hangs vertically when one side is released to allow the water tubs to pass, has meshes 4 inches square. It consists of galvanized steel wire ropes with a breaking strain of 21,000 lbs. per square inch of section. These ropes are formed of eighteen or twenty wires. The longitudinal ropes and the transverse ropes are bound together with galvanized wire to form the meshes. The net is formed with an edge formed of wire rope, with 133 wires about 1 inch diameter, and forms a rectangular net of 9 feet by 8 feet 8 inches. The net is suspended by bolts to two buntons placed below the hanging-on. Loops are formed on two opposite sides of the net, making the length of net, inclusive of loops, about 11 feet, and its weight about 290 lbs. The buntons to which it is suspended are 10 inches square and 12 feet 6 inches long. They are each fitted with five wrought-iron saddles, to which the loops of the net are attached by the wrought-iron bolts.

The cost of the net is from £20 to £25, and the cost of installation is about an equal amount.

This net has not been used in case of injury to workmen, but three accidents to material have taken place. Once the cage went too hard upon the keps, forced them back, and fell about 7 feet on to the net, which acted efficiently, and the damage was quickly repaired. Possibly in the absence of the net the rope would have been broken. On the second occasion, an empty tub and one of the cages, weighing about 6,600 lbs., fell into the net, from a height of 75 yards. Both tub and cage were comparatively uninjured, and the damage was cheaply repaired. On the third occasion, a full tub fell from bank, a distance of 350 yards, and the net was not seriously damaged.

M. W. B.

III.—NOTES OF PAPERS ON THE WORKING OF MINES, METALLURGY, ETC., FROM THE TRANSACTIONS OF FOREIGN SOCIETIES AND FOREIGN PUBLICATIONS.

THE ASSAYING OF ANTIMONY ORES.

Note sur l'Essai des Minerais d'Antimoine. *By* AD. CARNOT. *Annales des Mines*, 1892, *series* 9, *vol. i.*, *pages* 303-308.

The author states that on comparing the results of exact analysis with those of the ordinary method of assaying (*i.e.*, by fusion with sodium carbonate, charcoal, and strips of iron), he found that the error in the assay was rarely as little as 8 or 10 per cent., and was sometimes more than 20 and even 30 per cent. of the actual proportion of metal present in the ore. Considering the dry method objectionable on the score of the exceeding volatility of antimony, he bethought him of an entirely different process; this consists essentially in dissolving out the antimony from the finely-powdered ore by means of warm, concentrated, hydrochloric acid, precipitating the solution on a strip of tin, then washing, drying, and weighing the metallic precipitate. The foregoing process is immediately applicable to sulphides, as to oxides; the author converts the latter into the more easily treated sulphides, by heating the powdered ore in an atmosphere of dry, sulphuretted hydrogen. The presence of iron and zinc (the latter very rare in practice) in antimony ores does not affect the assay; almost the same may be said of arsenic. But as to lead, a metal which occurs in some varieties of antimony ore, the difficulty of its precipitation in the metallic state, together with the antimony, from the hydrochloric-acid solution, is met by the author in the following manner:—He heats the metallic precipitate to between 122 and 140 degs. Fahr. in a solution of yellow sodium sulphide (obtained by boiling ordinary sodium sulphide with flowers of sulphur). The antimony dissolves rapidly, and there remains a residue of lead sulphide, which is thereafter washed, dried, and weighed. In practice, the weight of the metallic lead is about $\frac{9}{10}$ths that of the sulphide.

O. S. E.

FUVEAU LIGNITE COAL-FIELD, FRANCE.

Mémoire sur le Bassin de Fuveau. *By* — OPPERMANN. *Bulletin de la Société de l'Industrie Minérale*, 1892, *series* 3, *vol. vi.*, *pages* 833-876, *and plates XVIII.-XXVII.*, *inclusive.*

The beds in this basin—situated in the Department of Bouches-du-Rhône—are contained in grey, marly limestone of Upper Cretaceous age. There are three beds worked, all included in strata to which the name of "Fuvélian" has been given—the grande mine, at the base of these strata; the quatre pans, some 164 feet higher; and the gros rocher, 33 feet above this. There are several other beds in the Fuvélian strata which are not worked, and also some unworkable beds in other series above and below.

The distances between the beds vary greatly in different places. Their thickness also varies considerably, that of the grande mine ranging from $3\frac{1}{2}$ to $11\frac{1}{2}$ feet,

of which 2½ to 10 feet is coal; the quatre pans bed varies in thickness from 4 to 4¾ feet, of which 1½ to 3½ feet is coal; and that of gros rocher from 1½ to 3¾ feet, 8 inches to 2½ feet being coal.

The total production of the basin in 1891 was 434,276 tons, practically all of which came from four mines in the hands of three companies.

Valdonne Colliery.—This is divided into three distinct portions by two great faults. Up to 1891, only the northern of these divisions was worked, and this is now nearly exhausted, having almost reached the western limit of the lignite basin, where it is cut off by the underlying Jurassic rocks. In this division the strike of the beds is north to south, and they dip west 1 in 8 in the upper, and somewhat less in the lower levels.

In view of the approaching exhaustion of this portion, a shaft has been sunk to a depth of 1,140 feet in the southern, or middle division, and this will in future be the main drainage-and-winding shaft for the whole mine. The remaining portion is almost entirely unworked.

Gréasque-Fuveau Colliery.—This is bounded on the south by the former. The continuity of the beds is interrupted only by two main faults, with a throw of 60 and 66 feet respectively, both being near the southern boundary. The portion of the concession south of the most northerly of these faults is alone now worked. Here the beds strike north, with an inclination of 1 in 6 to the west. The ground is full of minor faults, which are fewer in the district now abandoned to the north, where, however, they often become "moulières."

Trets Colliery.—This working is situated at the eastern extremity of the basin. Here the beds thin off, and the grande mine alone is workable. Its strike is north-west, with an inclination of 1 in 6 to the north-east. There are few faults, but the "moulières" are frequent.

Gardanne Colliery.—The same three beds are worked here as at Gréasque and Valdonne, with one other, known as the mauvaise mine. A great fault cuts off the beds to the north-east and west, but elsewhere, they are very regular, running north-west, and dipping 30 degs. to the south.

Throughout the basin exceptional difficulties have been encountered in the great quantity of water, which has necessitated heavy expenditure for adit-levels, dams, and pumping machinery. Figures are given of the quantities of water raised, cost of extraction, etc., of the Bouches-du-Rhône Company, from which it appears that 9,900,000,000 cubic metres of water were raised in 1886, at a cost amounting to 3s. 2d. on each ton of coal raised. From this cause considerable areas of coal have had to be abandoned, pending the construction of a long adit at sea-level.

Such faults as are not accompanied by throws, but which let out great quantities of water, are called by the miners "partens." These generally impoverish the coal on either side of the fissure, the sterile areas being termed "moulières." With reference to these, the region may be sub-divided as follows:—

1. The portions south of the Jean-Louis fault, with numerous fissures, rarely open, and often accompanied by throws.
2. The portion north of the fault, where throws are comparatively few, but with numerous "partens," with or without "moulières."

The "partens" tend to lessen in depth, and finish as mere cracks—the "moulières" also lessening proportionally. In the latter, at the point where the bed is cut by the fissure, the lignite is entirely replaced by a yellowish or blackish mud, diminishing with the distance from the fissure. The bed itself thins towards the "parten," the roof first becoming fissured, and finally is found completely fallen in.

The quantity of water underground is found to vary immediately with the amount of rainfall, and in times of drought the flow almost ceases. This does not apply to the Gréasque workings, which seem to be in communication with an extensive water-bearing stratum.

The author then discusses the geological character of the strata in respect of its bearing on the formation of the "moulières;" and finally describes the elaborate system of masonry-dams adopted in order to protect the workings from sudden flooding, etc. G. E. C.

THE DEEP ADIT-LEVEL IN THE FUVEAU LIGNITE COAL-FIELD, FRANCE.

Note sur le Tunnel à la Mer de quinze kilomètres. *By* — DOMAGE. *Bulletin de la Société de l'Industrie Minérale*, 1892, *series* 3, *vol. vi., pages* 899-919, *and plates XVIII., XXV.-XXVII.*

This is a description of the great adit, 9⅓ miles long, now being driven at sea-level to cope with the water difficulties in the Fuveau district. Particulars are given of the five principal adits hitherto driven, ranging in length from 820 to 3,565 yards. The history is related of the proposals for a sea-level adit made from time to time from 1859 downwards, culminating in 1879 in the present scheme, which was officially approved in 1889.

The rate of progress was, in 1891, 14·16 feet, and in 1892, 20·27 feet per 24 hours. The high speed is attributed partly to the use of the Berthet hand-perforator, and partly to the system on which the miners are paid, a premium being given for any progress beyond a certain rate per day. At first all shots in the face were fired simultaneously; subsequently, however, it was found advantageous to fire the middle holes first.

The *débris* is removed by means of an endless steel rope, 10 millimetres (0·39 inch) in diameter, carrying 150 tons per day. The ventilation is performed by a Ser ventilator, 23·6 inches in diameter, the waste air passing into the water-channel. This apparatus is calculated to serve for two-fifths of the distance, but the leakage which takes place makes it doubtful if it will serve for the whole. For this reason a ventilating shaft is to be sunk at a point about four miles from the entrance, and possibly another will be required farther in.

An Appendix is added giving details of method of firing, cost, etc., the latter being as follows for the advance heading (6 by 8·2 feet):—

	£	s.	d.
Labour	1	1	3
Explosives	1	1	2½
Coal, steel, and smith	0	1	2½
Total	£2	3	8

G. E. C.

THE VALDONNE COLLERIES (FUVEAU BASIN).

Note sur les Mines de Valdonne (Bouches-du-Rhône). *By* L. VALLA. *Bulletin de la Société de l'Industrie Minérale*, 1892, *series* 3, *vol. vi., pages* 877-897, *and plates XXIII.-XXVII.*

These mines comprise two concessions, known as Peypin and St. Savournin (north and south) with an area of 650 and 750 hectares (1,600 and 1,850 acres) respectively. They are situated at the south-western extremity of the Fuveau basin.

For details of the general constitution of the district the author refers to the paper by M. Oppermann in the same journal.* As there stated, the workings are divided into three separate zones by three main faults. The beds run north to south, dipping 10 to 20 degs. in the northern and 20 to 60 degs. in the southern part. There are seven beds, only the three following of which are worked :—

	Total Thickness. Ft.	In.	Useful Thickness. Ft.	In.
Grande mine	9	10	8	6
Quatre pans	5	3	3	11
Gros rocher	3	9	2	5

Immediately beneath the grande mine there is a bed of inferior coal 5 feet thick ; which, however, is rarely worked.

The lignite is brilliant black, with a conchoidal fracture ; it burns with a long flame and considerable smoke. Analysis shows :—

Moisture	6·00
Ash	4·40
Total carbon	60·65
Oxygen and nitrogen	25·87
Hydrogen	3·68

The calorific power is from 5,000 to 5,500 calories ; and the evaporating power 5 lbs. of water per 1lb. fuel.

The greater part of the extraction has hitherto been from the northern portion of the property, where all the hauling-machinery, etc., was situated. Recently, however, it has been transferred to the Armand shaft in the southern portion, where 600 to 700 tons are now raised from a depth of 1,140 feet per day of 9 hours. Owing to this change the production per man has risen from 10 to 16 cwt.

The beds are worked by galleries at an interval of about 660 feet on the dip. Of the grande mine about one-eighth has hitherto been lost in the pillars which were left behind. It is now thought possible to remove all the coal, stone pillars being built up in suitable spots, and walls built along the main roads. In working the quatre pans and gros rocher beds the deads are practically sufficient to fill up the goaves.

Powder is used for blasting both in coal and rock, the holes being bored by a hand-borer. The following comparative figures as to cost of boring galleries 2·50 by 2 metres (8 by 6½ feet) in the country-rock (limestone) are given :—

	Distance driven per 8 Hours. Feet.	Cost per Metre. £ s. d.
Powder	1·31 ...	2 2 0
Blasting-gelatine	4·26 ...	about the same.

The Armand shaft is divided into two compartments, for winding and pumping respectively. The winding is performed by two-decked steel cages, weighing 2 tons 7 cwts., guided on steel rails of 40 lbs. per yard. For winding, flat aloes-ropes are used, weighing about 25 lbs. per yard, working at one-tenth of the breaking strain. For raising the men, round steel-ropes are employed, weighing 3¼ lbs. per yard, working up to one-twentieth of the breaking strain.

The mines have always been subject, from time to time, to interruption from partial floodings. This, so far as it was due to water penetrating from the surface, has been minimized by establishing sheet-iron conduits in the stream-beds. The water underground now varies little from summer to winter. By this means, and also by the provision of masonry-dams underground, a great improvement has been effected in guarding against floods.

* *Trans. N. E. Inst.*, vol. xlii., page 416.

The drainage is effected partly by means of underground Tangye and rotary pumps, supplied with steam from the surface, and partly by Cornish and other engines at the surface. In future all the water will be raised by the underground engines from the Armand shaft, the present pumping-plant being reserved for emergencies only. This will give a total power of 2,861 gallons per minute, which should remove any danger from flooding in future.

The total cost of the works in connexion with the new Armand shaft was about £30,000, and it is expected to raise 160,000 tons of coal per annum. G. E. C.

PEAT IN TRANSYLVANIA.

Die Torflager der Siebenbürgischen Landestheile. *By* GEORG PRIMICS. *Mittheilungen aus dem Jahrbuche der Kön. Ungarischen Geologischen Anstalt*, 1892, *vol. x., pages* 3-24.

The following peat districts are described:—

(1) Magyar-Valko, province of Kolozs.—This district is estimated to yield 171,990 cubic yards of peat. The quality is very good; its lightness when dried would render it suitable for use in many industries. The hindrances to its profitable working are the great distance from railways, etc.

(2) Ponor, province of Alsó-Fehér.—This district is estimated to yield 390,000 cubic yards of peat. The working of this peat at present is not practicable, as it cannot be reached with waggons. It might, however, without very great difficulty, be sent to the neighbouring towns of Offenbanya and Felsö-Szolcsva. The above two peats belong to the class of high moor-peats.

(3) Maiótlaka, province of Kolozs, is estimated to contain 234,000 cubic yards of peat of good quality, and tolerably heavy when dried. It is a matured peat, water appears only in the deep parts of the cuttings, and would be no hindrance under a proper system of working. There are no reasonable hindrances to the working of this peat, and carriage will be easily effected, as a railway runs within 1½ miles, and a high road within ¾ mile.

(4) Szent-Agotha and Apátfalva, province of Nagy-Kürüllö.—This deposit is estimated to contain 1,300,000 cubic yards of a completely formed and dry peat. The quality is very good. It is situated near good roads of transport for local consumption. There is at present no railway near, but the projected local railway will run near the deposit.

(5) Szombatfalva, province of Udvarhely.—There are numerous mineral springs in the neighbourhood of this moss. The estimated yield is 37,440 cubic yards of good turf-peat (*Rasentorf*). The quality improves with depth. There are no difficulties in the way of working of this peat, as the water can be easily run off into the neighbouring burn, and a high road runs from the place to the Székelyudvarhely railway-station.

(6) Peat region of Middle Csik, along the Alt river:—

(*a*) The deposit between the town of Csik-Szereda and the parish of Zsögöd, is estimated to yield 1,690,000 cubic yards of pure peat, of good quality, and well matured. In many places there is much ground-water, but not sufficient to prevent the establishment of a large peat factory. The country road runs near, and the projected Székelyföld railway will touch the spot.

(*b*) The most important peat district in Transylvania is found in the parishes of Csik-Szereda, Taplocza, Csicso, and Mádéfalva. The estimated yield is 15,600,000 cubic yards of peat. The quality is very good, and peat production could be carried on on a large scale. The projected Székelyföld railway will run near this peat area along nearly its whole extent.

(7) Szerdahely, province of Szeban.—Three unimportant peat mosses, of which the total estimated yield is 1,909 cubic yards. Of the first, the peat is described as unfit for burning, and of the others as of middling quality. G. W. B.

ITALIAN FOSSIL FUELS.

Notizie sui combustibili fossili italiani. *By* P. Toso. *Appendix to the Rivista mineraria*, 1890, *Rome*, 1891.

Peat is extensively distributed as follows :—

(1) *Mountain Peat.*—Occurs in the Alps and Appenines, but hitherto has not been found of value.

(2) *Moraine Peat.*—Is found at Varese, Solferino, etc. These are the most important deposits, affording the greater part of the Italian peat produced. The author considers that they would keep up the supply for one hundred years.

(3) *Peat.*—Formed by damming back of water-courses *(Torbiere di sbarramento)*. The beds are found over large areas in Vicentino, Velino, near Rieti, etc. The peat is sometimes homogeneous and sometimes earthy. It is only worked at Arcugnano and Santa Croce.

(4) *Estuarine Peat.*—Occurs at the mouths of many of the rivers falling into the Adriatic and Mediterranean seas. It is the most abundant in Italy, but also the most impure and least remunerative to work. It is, however, worked at Codigoras for domestic use, for locomotives, etc.

It is estimated that the amount of peat produced in Italy is about 43,070 tons per annum. The necessity of drying it, and its light weight, are in the way of its increased production.

Woody Lignites.—Beds, over 6½ feet thick, are found at San Giovanni Valdarno, Morgnano, Santa Croce, Sant' Angelo, Branca, Aspra and Roccantica in Sabina, Casina and Castellina in Chianti, and Leffe, near Bergamo. Various deposits of less thickness are also described. The produce is about 331,000 tons per annum.

Pitchy Lignite (ligniti picee).—This fuel occurs in Tertiary rocks, older than the Pliocene. The following localities are mentioned :—Vicentino, Cadibona, Monterufoli, and Murlo. The mine of Montemassi is described in detail, it is estimated to contain 10,275,000 cubic feet of lignite still available. The principal stratum is about 25¼ feet thick. It is hoped to raise the annual production of this mine up to 50,000 tons. The quality of the lignite is good, and its calorific power is equal to 0·7 of that of Cardiff anthracite.

The total quantity of pitchy lignite is estimated at 8,570,000 tons, with an annual production of 33,160 tons.

Coal.—Coal itself is scarce in Italy, yet it does occur in the Carboniferous strata of that country. Its rarity is due to the earth-movements and consequent metamorphism which have taken place. Carboniferous strata with coal occur at the following places :—In the Val d'Aosta in the district of Cuneo, on the southern slope of the Maritime Alps, in the valleys of Tanoro, and of Bormida. Other Carboniferous strata containing no coal are also mentioned.

La Thuila in the Val d'Aosta has five or six seams under 1 foot 8 inches each. They are inclined at angles of 40 degs., and therefore difficult to work. The coal contains 25 per cent. of ash, and its heating power is only 5,000 calories.

There are mines at Monfieis, Acceglio, Calizzano, and Corongiu Cludinico (Triassic).

Various applications of the Italian lignites for gas-making, for engines, domestic use, and for coke-making, are described. G. W. B.

THE BOLEO COPPER-MINES, MEXICO.

Note sur les mines de cuivre du Boléo (Basse-Californie). *By* EDOUARD SALADIN. *Bulletin de la Société de l'Industrie Minérale*, 1892, *series* 3, *vol. vi., pages* 5-46 *and* 283-9, *and plates.*

The Boleo mines are situated on the western side of the gulf of California, about opposite to the port of Guaymas, which is the terminus of the Sonora railroad. They were discovered accidentally in 1868, by a farmer, who found rounded grains of green carbonate of copper in the sands of the Santa Rosalia rivulet. The first shipment of ore to Europe took place in 1872. The various local companies formed to work the deposits were, in 1885, united under the French Boleo Company, which erected water-jacket furnaces; the production in 1891 reaching 4,176 tons of copper from 76,000 of ore extracted.

The climate is stated to be healthy. The labour employed consists of Mexicans and Yaqui Indians, the latter work well, and are paid at from 1 to 1·75 dollars (Mexican) per day. The whites employed receive 15 to 20 francs per day.

The district is composed of stratified trachytic tuffs, slightly inclined north-east towards the sea, and resting to the south-west on a trachytic upheaval parallel to the coast. The central crest of the peninsula consists of old gneisses and mica-schists, raised probably by granite; but the interior, owing to the entire absence of drinking water, is little known.

The geological formation is divided into four series:—

(1) The upper beds, consisting of clays, gypsum, tuffs, and fossiliferous sandstone of Upper Miocene or Lower Pliocene age.

(2) The mineralized beds, consisting of four cupreous beds of varying thickness, interstratified between beds of conglomerate and tuff. Both these conglomerates and the cupreous beds appear to vary as to their contents with the distance from the source whence the detritic materials, of which they are composed, were derived.

(3) Lower beds. These mineralized beds rest unconformably upon a bed of brown fossiliferous dolomite of indeterminate age. Beneath this is—

(4) Trachyte, already referred to.

The cupreous beds are similar in general character, and can often only be distinguished stratigraphically. The contents are more or less brecciated, especially in the vicinity of the trachytic outliers, which sometimes interrupt the upper strata. They contain irregular zones of enrichment; and the contents, from a mineralogical point of view, are very varied. The fundamental gangue is a hydro-silicate of alumina, often very siliceous, and the predominant copper minerals are green and blue carbonates, black oxide, ferruginous crednerite, and various sulphides.

Near the outcrops the ores are entirely converted into carbonates, or into atacamite; while the iron pyrites is completely oxidized. Farther down, the carbonates are rare; the proportions of black oxide, and still more of sulphide, increase. The author questions whether the original deposit was in the form of sulphide, preferring to consider it to have been a mixture of this and oxide.

The first bed has been much eroded, and is of little industrial importance. The second is characterized by the comparative rarity of ferruginous matter and the abundance of silica in the gangue. The mineral is often of a granular nature, and is generally concentrated towards the floor. The thickness varies up to 3 feet.

The third bed varies in thickness from 0·20 to 2 or 3 metres (8 inches to 6 or 10 feet), and has been most extensively worked. It is not only richer, but the nature of the gangue enables a better smelting mixture to be made. Most of the ore is so finely disseminated in the gangue as to render dressing impossible.

The following analyses give an idea of the chemical composition :—

Moisture at 110°.	SiO_2.	FeO.	Al_2O_3.	MnO.	CaO.	MgO.	CuO.	Sulphur Total.	Loss by Heat.	Cu.
25·11	39·18	7·34	10·16	3·30	3·21	3·74	12·39	2·16	16·80	9·80
18·35	31·77	14·25	2·16	15·44	3·88	2·31	12·05	0·44	16·38	9·63
25·00	22·85	12·50	2·04	11·08	3·21	3·21	9·86	0·62	11·05	7·89
20·00	22·80	10·08	4·38	7·20	6·16	4·43	7·50	3·03	14·56	6·82

The moisture on extraction reaches 25 to 30 per cent., which on exposure to the sun is reduced to 15 per cent. The ore contains a little salt, which explains the corrosive action in the furnaces as arising from the giving off of a little hydrochloric acid.

The author considers the beds to be the product of re-arrangement of pre-existing beds of more equal richness, and thinks that the future of the poorer parts, now unworked, rests with the adoption of a wet chemical process of treatment.

There is nothing special to notice in the methods of working. Most of the selection is done in the mine, all the workings of which are above the water-level. In order to ensure regular supply, the ore as extracted is stored in ore-bins of large capacity, before being carried to the works ; which are connected with the mines by a 3-feet tramway of 30 kilometres (18½ miles) in length.

The ore is smelted entirely in water-jacket furnaces of ordinary type. It is stored in different bins, according to composition, so facilitating the attainment of a suitable smelting mixture. The smelter consists of 4 large rectangular and 2 smaller circular furnaces. Sea-water is used to circulate in the jacket, with fairly good results. The rectangular furnaces are of 80 to 100 tons, and the circular furnaces of 40 to 50 tons per day capacity. About 150 kilogrammes (330 lbs.) of good coke are used per ton of smelting mixture, the large furnaces being slightly more economical in this respect. The chief economy they afford is in labour, supervision, and cost of repairs.

The fines are compressed by the aid of a little water into balls by a machine on the Beer system ; they do not need drying before smelting. The blast is furnished by Baker and Root blowers. Notwithstanding the corrosive action of the sea-water used for the jacket, the furnaces usually give campaigns of over three months' duration.

No fluxes are required. The sulphur present is all taken up by the copper, forming matte, the rest being given as black copper. The slags contain on an average 1 per cent. copper.

The following analyses are given :—

MATTES.

Copper	96·250 ...	73·08
Iron	0·830 ...	4·24
Sulphur	0·767 ...	19·61
Manganese	traces ...	1·05
Arsenic and antimony	nil. ...	nil.
Loss, etc.	2·153 ...	2·02
	100·000	100·00

SLAG.

Silica	41·60
Alumina	10·26
Oxide of iron	15·57
Oxide of manganese	19·76
Lime	5·50
Magnesia	6·12
Oxide of copper	1·12
	99·93

At present English coke only is used, but it is not improbable that it may be replaced by anthracite, at a considerable saving.

The total number of workers employed is about 1,200. G. E. C.

THE COPPER REGION OF MICHIGAN.

BY F. B. PHELPS. *Engineering Magazine* (*New York*), *vol. iv., pages* 47-63.

The native copper deposits of the Keweenaw promontory occur as amygdaloid and conglomerate-beds interbedded with trap, resting on sandstone. The conglomerate-beds, ranging in width from 4 to 30 feet but fairly uniform, dip west, at an angle of from 32 to 52 degs. towards Lake Superior. The similar beds in the Isle Royale, an island in the lake 40 miles from the American shore, dip east; and it is surmised that they are the opposite outcrops of a continuous basin.

Ancient pits and implements prove mining here to be of great antiquity. The first modern workings on a large scale were in the northern region, from 1840 to 1850; but the exhaustion of the deposits, and the fall in prices, led to its almost complete abandonment. The great mining region is now in Houghton county, to the south, where the well-known Atlantic and Osceola (amygdaloid), Calumet and Hecla, and Tamarack (conglomerate) mines are situated. The mode of working is by inclined shafts on the foot-wall, with levels at regular intervals of 100 feet, the conditions for symmetrical mining being almost perfect.

At the Calumet and Hecla mines, the lode varies in width from 8 to 30 feet, with an average of 12 feet. The whole of the deposit is mined out, and the roof supported by very heavy timbering. The greatest depth reached at present is 3,850 feet on the incline, or about 2,350 feet vertical. The ore is stamped and dressed by hydraulic separators, the slimes being treated by round tables. The Tamarack Company works the ground to the west beyond the Calumet boundary, by means of deep vertical shafts. The first of these sunk cut the lode at a depth of 2,270 feet vertical.

The working of the amygdaloid beds is similar; the ore, however, is much softer.

G. E. C.

THE UNDERGROUND FIRE AT THE LAKE SUPERIOR MINE, ISHPEMING, MICHIGAN.

BY J. PARKE CHANNING. *The Engineering and Mining Journal* (*New York*), 1892, *vol. liii., page* 106.

The author has witnessed nine mine fires in the Lake Superior region, during the last eight years; five of which fires he had the opportunity of closely inspecting.

The conclusions arrived at by his experience are: that the best way to extinguish a mine fire is to cut off the supply of oxygen as quickly as possible, and to be very

careful not to open up too soon after the fire is supposed to be out. Steam is excellent, when it can be applied in the immediate vicinity of the combustion. He has not much faith in the use of artificial carbonic acid. While it is supposed to be heavier than air and to settle down to the bottom of the mine, it is, as generated from the tanks, very hot from the heat engendered by the chemical reaction producing it, and, consequently, it hangs around the collar of the shaft. A fierce fire will soon use up all the oxygen in a sealed-mine and then carbonic acid will be present in far larger quantities than can be generated by chemicals on the surface.

J. W.

OUTBURSTS OF CARBONIC ACID GAS AT THE ROCHEBELLE COLLIERIES, FRANCE.

Les dégagements d'acide carbonique aux mines de Rochebelle. By C. LANGE. *Bulletin de la Société de l'Industrie Minérale*, 1892, *series* 3, *vol. vi., pages* 1,141-1,180, *and plates XXXIII-XXXV.*

The mines comprise two distinct regions, in which carbonic acid and fire-damp are respectively given off. The presence of the carbonic acid was observed from the first, but it did not give rise to any serious danger up to 1880, and mechanical ventilation was not thought necessary.

Besides the smell and sour taste, a sensation of warmth is felt in the portion of the body immersed in the gas; so that its presence is easily detected. Like fire-damp, it is enclosed in the pores of the coal, but with great irregularity; and almost certainly it did not originate *in situ*.

The mines occupy the southern portion of the Gardon basin, the beds being included in the lower system of the Gard coal-field. The concession is included between the great fault of the Cévennes to the east and the ancient schists of Mont Cabane to the west. Near the former the beds are much bent and fissured. To the north, in the valley of the Gardon, they are covered by Secondary measures (Trias and Lias): at the base of the Lias important deposits of iron pyrites and oxide of iron are worked.

The workings are classed into three groups, those of Rochebelle, Cendras, and Fontanes, at all of which more or less carbonic acid is present. At Fontanes a schistose formation predominates, which is less fissured than the compact sandstones at Rochebelle. Probably for this reason, the carbonic acid at the latter is contained in the fissures, while at Fontanes it seems to be enclosed in the coal. It appears to be diminishing in quantity at Rochebelle, and to be replaced by fire-damp—which, however, was not observed until 1886. The author considers the carbonic acid to be of volcanic origin, and to have been brought in by the Soulier fissures, which also conveyed the mineralized solutions which gave rise to the pyrites deposits.

He next describes in detail the various outbursts of carbonic acid. These at Fontanes were so sudden as to be practically explosions. They gave out enormous quantities of gas, and threw down a great deal of coal—amounting at times to hundreds of tons. In several cases the loss of life was serious.

At Rochebelle the outbursts were produced with less force; and instead of coming from the solid coal, they proceeded from open fissures, full of water and carbonic acid. The walls of these fissures are found to be covered with quartz-crystals with liquid enclosures, blende, galena, and pyrites.

In every case the outbursts have been preceded by one or more of the following symptoms:—Heaviness of the atmosphere; frequent dull explosions; pressure in the boreholes; detachment of plates of coal, with decrepitation, and changes in the physical state of the coal.

The force of the outbursts appears to increase with the depth.

Experience has demonstrated that the most useful precautions to be observed when the presence of the gas is suspected are:—(1) Arrangements for supplementing ventilation; and (2) entire suppression of the use of the pick, and working solely by means of shot-firing: the workmen to retire behind solid barriers at the moment of firing.

It was found that boreholes were quite useless to drain the gas from the coal.

G. E. C.

ORIGIN AND DISTRIBUTION OF GOLD AND PLATINUM, NORTH COAST BEACHES, NEW SOUTH WALES.

By J. W. Archibald. *Australian Mining Standard, vol. ix., pages* 193 *and* 220.

The gold in these deposits occurs in seams of black sand; it is always fine, and associated with the rare metals of the platinum group. The black sand is found only at the high-water line, or in the sand terrace forming the foreshore; the former being in many cases naturally concentrated from the latter. It usually presents no difficulty in amalgamation; south of the Evans river, however, part of the free gold, though bright, can only be saved in the blanketings.

In the vicinity rock-formations are met with, carrying the various metals found in the sands. A sample of the blanketings from the latter contained:—Gold 27 ounces, platinum 17 ounces, and osmiridium 140 ounces per ton. No free gold was visible, this having been extracted by amalgamation. Rock-samples from near by gave:—Gold 10 dwts., platinum 9 dwts., and osmiridium 21 dwts. per ton. When concentrated, the residue was heavy brown sand, neither gold nor platinum being visible. As an example of these formations the "Iron Gates" of the Evans river is cited. This is a gossan deposit, about 1 chain wide, lying parallel with the coast, and imbedded in the coal-formation, the sedimentary rocks appearing unaltered at the contact. This deposit carries 1 to 2 ounces per ton of the combined rare metals.

Many other such outcrops occur in the neighbourhood, and the natural inference is that the beach-sands are derived from them, their subsequent distribution being effected by sea-action.

G. E. C.

THE MOUNT MORGAN MINE, QUEENSLAND.

By T. A. Rickard. *Transactions of the American Institute of Mining Engineers,* 1891, *vol. xx., pages* 133-154.

Mount Morgan mine is situated 26 miles south west from Rockhampton, in central Queensland. Up to November 30th, 1890, £2,358,333 had been paid in dividends, the total gold obtained being 756,042 ounces, worth £3,121,741. The property consists of the original selection of 640 acres taken up for grazing purposes in 1873 by Mr. Donald Gordon. Becoming acquainted with Messrs. Morgan Brothers, who also held land in the district, he showed them one day a piece of gold-bearing quartz which he had picked up in Mundic creek. For a consideration, Mr. Gordon agreed to indicate to them the locality of the find. On the hill, overlooking the creek, he showed them the siliceous ironstone, some of which can still be seen cropping out on the north-eastern slope. The stone carried visible gold. They found, by sending samples to Sydney, that it was even richer than they had imagined. So they purchased Mr. Gordon's holding at £1 per acre. The Messrs. Morgans, later on, sold first a part, then the whole of their interest in the mine. In 1886 a company

was formed with a capital of 1,000,000 shares of £1 each. These shares rose toward the end of 1888 to £17 5s., giving the mine a market value of £17,250,000.

The author gives a description of the workings.

Examination of the many varieties of ore shows that while there may be a great difference in outward appearance due to coatings of multi-coloured oxides, the ore is always substantially quartzose.

The origin of the Mount Morgan ore-deposit has been the theme of much controversy. The three chief theories (each of which the author discusses at length) are :— 1st, That the deposit is that of a geyser ; 2nd, that it is an auriferous zone traversed by a series of quartz-veins of auriferous mundic ; and 3rd, that it is the decomposed cap of a large pyrites lode.

Mr. Rickard considers that a period of dynamic disturbances is indicated by the intrusions of dolerite, which, by extreme metamorphism, might have changed a dolomite into the serpentine we now see, would have indurated the shales so that they are scarcely to be distinguished from the crystalline rocks ; and would also, accompanied by chemical alteration, change a ferruginous red sandstone into a bluish-grey, highly pyritiferous quartzite. Approaching the surface, the same energy would be expended in the fracturing of the quartzite and the graywacke ; the intrusive dolerite would rise through the fissures in the shattered rocks, forming dykes, which, meeting a silico-felspathic granular rock (the graywacke), would give it a semi-crystalline character. The sandstone would similarly be vitrified. Later movements would result in the further intersection of this part of the district by the numerous dykes, the decomposed remains of which are now to be seen ramifying through the deposit. Those gradual chemical interchanges would take place which resulted in the alteration of the shattered country-rock, and its becoming a portion of the gangue enclosing the auriferous material, which was then, or at a later time, deposited. In process of time, subaërial denudation removed the sandstone, which now is only to be seen on the further summits of the neighbouring hills. Atmospheric agency continued to carve away the less siliceous and less porous portions of the country surrounding the deposit, until Mount Morgan, owing to the pervious, quartzose nature of its crest, remained as a low hill in an undulating country.

Since writing his paper, the author has visited and examined the equally famous Broken Hill silver deposit, and the latter confirms a statement he makes that the material of the Mount Morgan ore could be duplicated elsewhere. At Mount Morgan all the conditions generally considered favourable to ore deposition were present in a marked degree. A mass of country-rock, whether graywacke or sandstone, consisting of granules of quartz, held together by a felspathic cement, becomes fractured by the intrusion of a series of dykes, which form a ready passage for the flow of mineral solutions. Such a rock would readily lend itself to alteration, and would be well fitted to receive a mineral deposit. Later dynamic action produced a further metamorphism of the surrounding rocks, followed or accompanied by the intersection of the already shattered rocks by another series of dykes, which reopened a passage for the underground waters. In this case it is not necessary to go far for the source of the gold. The large mass of decomposed pyritic quartzite, though the pyrites contains but a trace of the precious metal, is more than sufficient to account for the wealth that has been uncovered.

Finally, says Mr. Rickard, there is no need to offer any particular reason for the unusual richness of the deposit, beyond the eminently favourable condition of the original rock, the numerous channels offered to percolating waters, and the close proximity both of the sources of the gold and the usual precipitants. J. W.

INFUSORIAL EARTH.

La Gloria Infusorit. *By* G. PETIT. *Génie Civil*, 1892, *vol. xxii., page* 77.

The deposits of infusorial earth, to which the name of *Gloria infusorit* has been applied, at Celle in Hanover, and Klicken in Anhalt, are of great extent. They occur in basins, often attaining a thickness of 66 feet (20 metres), from 3 to 15 feet (1 to 5 metres) below the sandy surface. They present three distinct qualities, white at the bottom, then grey, and green at the top; this latter resulting from the vegetable products which have worked in from the surface, and which often form 40 per cent. of the whole.

The main use at present is in the manufacture of dynamite; but it is also suited for all purposes where a good non-conductor of heat is required. G. E. C.

JADE IN UPPER BURMA.

Note on the occurrence of Jadeite in Upper Burma. *By* DR. FRITZ NOETLING (*with a map*). *Records of the Geological Survey of India*, 1893, *vol. xxvi., pages* 26-31.

The chief interest of this paper is the description of the manner of occurrence of jade. It is found forming the central portion of a dark eruptive rock closely resembling serpentine, and separated from the outer rock by a band of soft clayey serpentine. The rocks through which the serpentine has been erupted are probably of Miocene age. G. W. B.

IMPROVEMENTS IN COPPER SMELTING.

By E. D. PETERS, JUN. *Australian Mining Standard, vol. ix., pages* 276 *and* 288.

Pyritic smelting, *i.e.*, smelting pyrites by the heat generated by the combustion of the contained sulphur, is stated to be successfully performed in the United States, at Leadville, Kokono, and Boulder, by heating the air-blast. This concentrates the temperature in the lower part of the furnace, so preventing the ore in the upper part from becoming soft and clogging. An ordinary water-jacket blast-furnace is used, and the smelting commenced as usual. After it is working well the proportion of coke is lessened, until with pure pyritic ores it may even be dispensed with entirely. At Boulder, where the sulphur in the charge averages 20 per cent., about 2½ per cent. of coke is added, as against 15 per cent. formerly required. A large proportion of fines is of course a hindrance, and over 40 per cent. would render the method inapplicable; as also would over 5 per cent. of lead or zinc sulphides.

The rest of the paper contains a description of the Manhès process of Bessemerizing copper mattes. G. E. C.

MANGANESE IN THE UNITED STATES.

Manganese. *By* R. A. F. PENROSE, JUN. *The Engineering and Mining Journal* (*New York*) 1892, *vol. liii., pages* 38-39.

Production.—In 1891, the production of manganese ore proper, independently of manganiferous iron ores, manganiferous silver ores, and manganiferous zinc ores, has been probably less than 20,000 tons. The production for 1889, was 23,927 tons, whilst the largest annual output in the United States was in 1887, when 34,524 tons were mined.

Most of the manganese ore of the United States is produced in the States of Virginia, Georgia, and Arkansas, mentioned in the order of the quantity of ore produced; while smaller amounts are derived from Leadville, Colorado; San Joaquin County, California, and the Lake Superior region of Michigan. The old manganese workings at Brandon, Chittenden and South Wallingford, Vermont, were idle in 1891.

Methods of Mining, etc.—The largest manganese deposits in the United States—those of Virginia, Georgia, and Arkansas—occur as irregular nodules, layers, and pockets in clay. They are usually worked by shafts and drifts, and the loose character of the ground frequently necessitates expensive timbering. In some cases large open-cuts have been worked successfully. In Colorado and California the deposits occur in solid rock, and the ore is obtained by the ordinary process of mining.

The ore in the clay is usually washed before shipment in an ordinary iron-ore washer, as in Virginia and Arkansas, or in revolving perforated cylinders as in Georgia. The smaller ore is also sometimes cleansed by jigging.

Uses of Manganese.—Over nine-tenths of the manganese production of the United States is consumed in the manufacture of ferro-manganese and spiegeleisen for use in steel-making. The rest, usually the higher grades from Virginia and some from Georgia, is consumed in the manufacture of bromine, in clarifying glass, in making manganese bronze, silver bronze, and other alloys, besides in small quantities for numerous other manufacturing and chemical purposes. J. W.

THE RUSSELL PROCESS AT THE SOMBRERETE MILL.

By E. H. Russell. *The Engineering and Mining Journal (New York)*, 1891, *vol. li., page* 140.

The Russell process (English patents, 5431, March, 1884, and 6102, April, 1888), has for its object the recovery of silver and gold existing in ores as metallic silver, as sulphides of silver or gold, or as compounds of silver with antimony or arsenic, and which cannot be extracted by leaching with sodium hyposulphite alone. The leaching solution employed in the Russell process consists of an ordinary hyposulphite solution to which has been added a soluble compound, preferably the sulphate of copper. A double salt of cuprous sodium hyposulphite is thus formed in which the copper can be replaced by silver, the eliminated copper taking the place of the silver in the compound decomposed by the solution. The solution is made by adding 1 to 3 per cent. of the copper salt to a 1 to 5 per cent. solution of hyposulphite of soda. The gold and silver are recovered by precipitation as sulphides. According to the newer method, protected in 1888, the copper sulphate is first mixed with the crushed ore, so as to replace by cupric hydrate any of the injurious metallic hydrates which may be present in the charge. When this neutralizing action is complete, the usual sodium hyposulphite solution is added, this reacts with the copper, already present in excess, to form the cuprous sodium hyposulphite.

The series of experiments reported on were instituted with a view to determine the best mill methods for the treatment of Sombrerete ore, and to decide what changes might be necessary in the new plant.

The experiments were confined to ore from the San Francisco mine, an approximate analysis of which was as follows:—

FeS_2, 40 per cent; ZnS, 12 per cent PbS, 13 per cent, and insoluble residue, 34 per cent.

The blende assayed 80 ounces of silver per ton; the galena 70 ounces, and the pyrites 35 ounces.

The leaching work done in the mill was so thorough that the average apparent mill extraction, in the whole 127 charges (40 tons each) treated, exactly equalled the result of the assay-office leaching test on the samples of these charges, and for the last 10 of the 13 mill runs, exceeded the assay-office results by 1·4 per cent. These experiments at the Sombrerete mill ran over 93 days, 4,935 tons of ore being crushed.

The whole series of experiments was divided into three sections, according to the general method pursued in the roasting, as follows:—

(1) *Furnace Roasting without an Air Blast and without a Preliminary Field-roasting.*—The following results were obtained by this method of working:—

Per Cent. of San Francisco Ore used.	Mesh of Screen for Ore.	Mesh of Screen for Salt.	Per Cent. of Salt.	Per Cent. Extraction by Extra Solution in Assay Office.
100	60	30	17	92·3
100	60	0	0	71·4
76	30	12	12	74·6
90	24	12	14	71·1
100	24	12	12	54·5

(2) *Furnace Roasting with Air Blast in the Furnace, but without a Preliminary Heap-roasting.*—With this method of roasting, and with an ore containing only 17 per cent. of sulphur, crushed through a 24-mesh screen and roasted with 11·5 per cent. of salt added to the raw ore, the actual mill extraction based on raw-ore weights and values was 79·8, or if based on roasted-ore weights and values 95 per cent. All the ore treated under this section was crushed by rolls. The result showed that for roasting, except after fine crushing, the sulphur contents of the ore should be reduced.

(3) *Furnace Roasting after a Preliminary Heap-roasting.*—With the object of reducing the sulphur contents of the ore, heap-roasting was adopted. The piles were built as follows:—Trenches about 1 foot in depth and width were dug about 2 feet apart. Similar trenches were made at right angles across these. All were covered with old grate-bars or flat stones, laid an inch or two apart. The trenches, under the grate-bars, were then filled with kindlings and the ore was piled on top to a height of from 6 to 12 feet. The following features about the heap-roasting were worthy of note:—No silver was lost; the amount of fuel required was about one-half cord of wood per 500 tons of ore; and the piles burned rapidly, a 300 tons heap only requiring four to five days. With such rapid burning, however, the interior of the pile sintered. This was remedied by covering the pile with fine material, to cause slower combustion. All the field-roasting was done at the mine, scarcely any extra handling of the ore being required. The apparent extra expense of heap-roasting was about 15 cents per ton. Really, there was no extra expense as, by heap-roasting, the drying of the ore at the mill, which was otherwise necessary, and which cost 25 to 30 cents per ton, was avoided. During this section of experiments the salt used in roasting averaged 8·7 per cent. of which 1 per cent. was added to the raw ore and the remainder thrown into the shaft and return flue of the furnace on top of the roasted ore.

The average number of pounds of chemicals used per ton in leaching was:—Hyposulphite, 1·8; bluestone (copper sulphate), 6·3; sodium carbonate, 6·3; caustic soda, 4·0; and sulphur, 2·6. The strength of the stock solution was 1·5 per cent. of hyposulphite, and of the extra solution, 0·70 per cent. of bluestone.

The average time of charging vats (50 tons) was 8 hours: of first washing, 12 hours; leaching and second washing, 46 hours; sluicing-out tailings, 1½ hours; or a total of 67½ hours.

The average loss of silver by dust from machinery and furnace, and by volatilization in roasting, during the experiments of this section was 3·6 per cent. of the raw ore weights and values, that of the last run being 1·3 per cent. The actual mill clean-up, based on raw ore weights and values for all the experiments of this section was 86·3 per cent., that of the last run being 90·9 per cent., or, if based on roasted ore weights and values, 93·0 per cent.

Summary.—The changes in mill methods and the results obtained in the treatment of Sombrerete ore, containing about 66 per cent. sulphides, are summarized as follows :—

All drying of the ore at the mill was discarded : the ore crushing was changed from 60-mesh to 6-mesh ; the fineness of salt used was changed from 30-mesh to using it uncrushed and undried : the method of using it was changed from adding it to the raw ore to adding it to the roasted ore ; the amount of salt used was reduced to 6·5 per cent. ; the loss of silver, as dust and by volatilization, was reduced to 1·3 per cent. ; the results of leaching in the mill equalled and exceeded the results which could be obtained by assay-office leaching tests on the same roasted ore. The results attained on all heap-roasted ore averaged 86·3 per cent. actual extraction, based on raw-ore weights and values, or 90·9 on the last run, or 93 per cent., based on roasted-ore weights and values ; all of which were about such as would be obtained in the same running time and with the same amount of ore in any new mill. The results at the Sombrerete mill compared with those of the Marsac mill (Daly Mining Company) were as follows :—

	Silver. Ounces per Ton.	Gold. Value per Ton.	Per Cent. Silver extracted.	Per Cent. Gold extracted.	Cost of Mining per Ton.	Cost of Milling per Ton.
Sombrerete	40·00	$2·25	83·0	50·0	$6·79	$6·48
Daly ...	43·40	1·35	89·2	41·5	5·50	6·40

J. W.

THE PRACTICAL CHLORINATION OF GOLD-ORES AND THE PRECIPITATION OF GOLD FROM SOLUTION.

By John E. Rothwell. *The Engineering and Mining Journal* (*New York*), 1891, *vol. li., pages* 165-166.

The author gives records from successful working experience on a large scale. The chief objection to a plant of a capacity of 50 tons, or more, per 24 hours, for the Plattner (chlorination) process is its large size and the length of time required for a single operation. The problem to be solved by the engineer, therefore, in handling low-grade ores that will not concentrate, is to find a process that will treat large quantities of ores quickly, cheaply, and with as little interruption as possible. In the reduction works of which this article treats, the ore was crushed dry, in Blake and Gates crushers and in two sets of Krom rolls, roasted in Brückner furnaces of three tons' capacity, and chlorinated in barrels of three and four tons' capacity.

Crushing.—Preliminary experiments must be made to ascertain how coarse the ore may be crushed, so as to yield the best results. The pulp for the best leaching must be in granular condition and carry as small a percentage of dust or slimes as possible. Rolls, properly managed, are especially well adapted for this purpose.

Chlorination.—The barrel is made to serve also as the washing and leaching vessel by placing a supporting diaphragm, for a filtering medium, to form the chord of an arc of the circle of the barrel. This filter (or diaphragm) is made of plates, corrugated as for an ordinary filter-press, and perforated with holes every four, or six inches square. The plates are supported on segments which are bolted to the shell ; on the top of the corrugated plates is placed the filtering medium, an open-woven

asbestos cloth. Two methods are described for charging the pulp and chemicals—bleaching powder and sulphuric acid. When the chlorination is completed, the barrel is stopped so that the filter is left in a horizontal position, a hose is attached to one of the outlet pipes and conducts the solution to a reservoir-tank.

The method of leaching is described, the washing being conducted so as to prevent escape of chlorine gas. The solution runs into a covered tank, from which a fan exhausts into the atmosphere, outside of the building. The water used was 100 gallons per ton of ore for chlorination, and 120 gallons more for leaching. By using the washings over again, the volume of solution to be precipitated is kept down to 120 gallons per ton of ore.

The advantages of this method are said to be:—Considerable saving of skilled labour, freedom of the building from chlorine gas, control over leaching of charge, and the small amount and simplicity of machinery for amount of work done. One intelligent man and a helper can look after the charging, leaching, and discharging of three barrels.

Precipitation.—The writer has found hydrogen sulphide gas, generated from paraffin and sulphur, or from iron sulphide and sulphuric acid, to be the cheapest and most satisfactory precipitant. The hydrogen sulphide gas is forced through the solution by a small air-pump, which at the same time forces air through, this agitates the solution and expels the free chlorine gas. Precipitation is thus rapidly effected and the precipitate settles well in a flocculent form. The asbestos filter cloth can be changed in about 1½ hours, and will last for over 100 charges.

The wear and tear of the plates, grating and lining of barrel is said to be very slight indeed. J. W.

THE CHLORINATION OF GOLD-ORES.

By J. H. Burfeind. *The Engineering and Mining Journal* (*New York*), 1891, *vol. li., page* 446.

An outline of the Plattner process is given, as worked at Douglas Island, Alaska, by the Alaska Treadwell Gold Mining Company, which works turn out more gold than any other works in the United States.

Firstly, with regard to precipitants, the author finds none equal to ferrous sulphate for general use. Its preparation is simple, requiring no machinery; the material is cheap and always at hand; its cost is less than any of the others offered to replace it; its application is simple and its action entirely satisfactory.

The main objection to the use of hydrogen sulphide is that it precipitates copper, as this metal is rarely absent, and the separation of the sulphides is always difficult.

The works at Douglas Island have been in operation about seven years. The material treated is the sulphides collected by the Frue vanners in the stamp mill. They contain on an average over 40 per cent. of sulphur, mostly in iron pyrites, although of late a good deal of copper pyrites has made its appearance. The gangue is quartzose, with from 2 per cent. to 5 per cent. of calcite.

The first improvement made was to replace the Brückner with the automatic Spence furnace. The latter did not answer at all until changed from a muffle to a reverberatory. Six of them were built at a large expense. The cost of roasting in them was less than half that in the Brückner; but their capacity was small and fuel consumption large. An old-style reverberatory furnace was also added and such satisfactory results were obtained by it that the company decided to use them entirely. The six double Spence furnaces were thrown out, and now the works treat nearly twice the amount of material with only four reverberatory furnaces

These four furnaces, built after the plan of Mr. H. Stansfield, are 13 feet by 65 feet inside measurement, and handle about 20 tons of material daily at a cost much less than could be done with the Spence furnace.

The roasted ore is removed from the pit under the furnace, spread on the floor to cool somewhat, then wetted sufficiently, sifted into vats, each holding about 4½ tons, and the chlorine gas is put on. Four hours, on an average, are sufficient for the gassing operation; after which the vat is allowed to stand 20 to 30 hours.

The leaching usually takes 12 hours. For convenience, and to avoid breakage of filters, etc., the solution is run into intermediate tanks and thence to the precipitating vats in which is already placed the necessary amount of the precipitant, Fe SO_4. The precipitation is complete by the time the vat is run full. The solution is then stirred briskly for a few moments and left to settle for 18 to 24 hours. The supernatant liquor is then drawn off and allowed to go to a large filter. Tests show that this waste liquor, when down, contains gold still in suspension, equal to 23 to 25 cents per ton of material treated, and this is entirely saved by the filter. A clean-up is made twice a month; as, if only once a month, the amount of gold is too much for one man to handle. The drying and melting of, say, 12,000 dollars (£2,400) can easily be done by one man in one day. The dried gold, from the clean up, is melted into bars with a little borax.

A chlorination-vat holding about 4½ tons of roasted material costs 50 dollars (£10), and lasts fully three years without any repairs. The filter in it costs only the price of a few gunny sacks, will last six months, and needs no attention during that time. The other vats will last a lifetime and not need more than an occasional hoop. There is not a piece of machinery in the works, an advantage that is fully understood by a practical chlorinating man. The real difficulty which is universally met is in the roasting of the material. Most of the ores now treated by chlorination are roasted with salt. Whilst intelligent operators have succeeded in reducing the loss of gold, which occurs by volatilization very largely when salt is used in roasting, it is still very large and occasionally will increase to an alarming extent without any apparent cause. The loss from this source amounts to an enormous annual aggregate, and is often the reason why well-arranged and well-conducted works prove financial failures. J. W.

LEAD-ORES OF MAZARRÓN, SPAIN.

Criaderos Metaliferos de Mazarrón. *By* F. B. Villasante. *Revista Minera, vol. xliii., pages* 145, 155, 163, and 169.

The mineral deposits of Mazarrón are of two classes—veins and masses. To the first class belong the lead veins, which form the wealth of the district, the occasional veins containing copper ores being of no importance.

The lead veins contain more or less argentiferous galena, often intimately mixed with blende, and this latter is sometimes very largely in excess. The containing rock is always trachyte, and when this disappears in depth the fissure which has formed the lode continues, but is always barren.

The galena is either laminar, radiated or granular; in the first case being usually richer in lead and in the second in silver. The richness is very variable, in some cases being 65 to 80 per cent. of metallic lead—in others not more than 10 to 12 per cent. The average may be taken at 30 per cent., which by mechanical concentration may be raised to 50 per cent. The ley in silver varies from 1 ounce and 2 ounces per quintal to 3½ ounces, 2 ounces being very common; but in one district there are veins of carbonate very poor in lead, but with 6 ounces of silver as the average.

The direction of the more profitable lodes is pretty constantly north and south, within a few degrees. There are lodes running east and west, but these, with few exceptions, are barren. The inclination varies from 45 degs. to vertical, dipping more generally to the east. Changes of inclination are, in many important lodes, very frequent. The width also varies greatly, from 8 inches to 16 feet (0·20 of a metre to 5 metres). Geodes (vughs) are infrequent, and are generally small fissures or irregular cavities full of earthy dolomite.

The mineral constituents of the lodes vary greatly, not only at different depths, but even at the same level. Near the surface the galena almost completely disappears, being replaced by carbonates, phosphates, and arseniates of lead, and there is also a great deal of brown hæmatite. The superficial decomposition of the trachytes by the metalliferous solutions permeating the fissures has given rise to the formation of alunite, which forms the best evidence of the existence of a lode.

The richness of the veins has been observed to be greatest in their most vertical portions, in accordance with the theory of Moissenet. It has hitherto been considered that the richness in silver diminishes with depth, but two important discoveries which have been recently made do not favour that assumption.

Not all the lodes reach the surface, although the more important do; the remainder may be considered disconnected branches from these. The course is, as a rule, exceedingly irregular, and even when the fissure continues for a long distance, the width and contents are extremely variable.

In the succeeding numbers the author describes the various mines in detail.

In the Fortuna district the two dominant directions of the lodes are north 15 degs. east and north-west. The inclination is always nearly vertical, with a slight tendency to dip north-west. The thickness can hardly be determined, owing to the small depth hitherto attained, but may perhaps be put at about 3 feet. The minerals present and the ley in silver vary greatly, but in the former group the general rule is for the proportion to be 9 to 10 ozs. silver per quintal of lead, in the latter group not more than 4 ozs. The outcrops are very numerous, and generally traces of old workings.

With reference to the iron deposits, in the Santa Justina mine they consist of superficial pockets intercalated in Tertiary limestones. The average percentage is 50 per cent., with a little manganese.

The Vulcano mine is an east-and-west lode dipping north, and for a time yielded 1,000 tons of ore monthly. At a depth of 328 feet (100 metres), however, it disappeared, and its continuation could not be found. G. E. C.

MICA MINES OF CAROLINA, U.S.A.

Mica and the Mica Mines. *By* C. Hanford Henderson. *Popular Science Monthly* (*New York*), *vol. xli., pages* 652-665.

The mica mines of the western part of North Carolina are described. The country rock is Huronian or Laurentian granite and gneiss, dipping at an angle of 45 degs. or more. Commonly the veins occur in a fine-grained black gneiss known locally as "slate." They usually dip with the bedding, but sometimes cut across it; and vary from under an inch to 10 or 12 feet thick. The contrast between vein-stuff and country is very striking, the felspar in the former giving it a glistening white appearance. The vein-filling consists essentially of mica, quartz and felspar, very coarsely crystalline, the mica having crystallized out first.

The mines were first found by ancient trenches on the veins, where the "old men" had worked with stone implements. They were first tried for silver in the idea that this was what the ancient workings had been for, but were naturally found barren. Subsequently the mica on the waste heaps attracted attention, and led to systematic mining.

The mica as extracted is in the form of rough hexagonal prisms, varying in colour from silver grey and green to a rich brown—the latter, known as "rum" mica, often fetching a higher price. The arrangements for working and transport are of the most primitive description.

The prisms are split up into thin plates and cut to various patterns.

Occasionally the blocks are very large, when they become valuable. One is reported to have been 6 feet long by 3 feet wide.

The cut mica, after careful trimming, is put up in pound packages and marketed. The waste is very great, 100 lbs. of block yielding only about 15 lbs. of cut mica. The scrap is screened to free it from sand, etc.; it is then ground and used in various industries. G. E. C.

THE MAGNETIC ORE-CONCENTRATION WORKS AT MAIERN, TIROL.

Die Erzaufbereitung in Maiern mit besonderer Berücksichtigung der Elektro-magnetischen Extraction. By Josef Billek. *Oesterreichische Zeitschrift für Berg-und Hüttenwesen*, 1893, *vol. xli., pages* 39-44.

This mill was erected to concentrate galena, blende, and iron ores. The separation of these minerals is effected partly by the ordinary wet process and partly by magnetic means. The zinc blende is found intimately associated with siderite (carbonate of iron) which minerals having very similar specific gravities, cannot be separated by the wet method, the specific gravities being zinc blende, 4·0; and siderite, 3·6. A system of magnetic separation is therefore adopted, and the ore is roasted so as to change the siderite into magnetic oxide of iron.

The ore from the mine is separated into three classes as follows:—1. Galena and blende ore, rough ore (*erzwände*). 2. Smalls (*grubenklein*). 3. Blende ores (*blende mittelerze*). The blende, which constitutes the main portion of the production, is found in two varieties; soft coarsely crystallized blende, containing on an average 57 per cent. of zinc, and hard blende, with which siderite, magnetite, and galena are associated, containing 40 per cent. of zinc.

The ore as it comes from the mine is first passed through a stone-crusher, which crushes in 10 hours from 45 to 70 tons to an average size of 1·77 inches (45 millimetres). The larger size, above 0·63 inch (16 millimetres) is hand-sorted, and the product may be quantitively classified as follows:—

	Per Cent.
Best blende	1·5 to 2
Lead ore	3·5 „ 4
Blende ore	70 „ 80
Waste	20 „ 25

The cost of hand-picking is 35 to 50 kreuzer (7d. to 10d.) per ton of crude ore. After the lump ore has been hand-picked, the ores are farther reduced by rolls, and are sorted according to size by means of revolving sizing-drums preparatory to jigging. The ores are sized into particles from 0·04 to 1 inch (1 to 25 millimetres).

The jigs treat per 10 hours on an average as follows:—Fine jigs, 2½ tons of ore; coarse jigs, 7 tons of ore.

In order to render the siderite magnetic, the mixed blende and iron ore is roasted in kilns, which have an inside width of 3·28 feet (1 metre), length of 6·56 feet (2 metres), and height of 7·54 feet (3·2 metres). The carbonate of iron ($Fe\ CO_3$) is changed into oxide of iron ($Fe_3\ O_4$). Each kiln roasts $7\frac{1}{2}$ tons in 24 hours. After the first heating of the furnace no fuel is required, as the combustion is kept up by the sulphur in the ore.

The roasted ore is crushed by rolls, of which there are five pairs, each having a diameter of 18·9 inches (48 centimetres), and a width of 12·6 inches (32 centimetres) and making 30 revolutions per minute. The ore is crushed to four sizes, 0 to 0·02, 0·04, 0·08, and 0·12 inch (0 to 0·5 millimetre, 1, 2, and 3 millimetres). The dust made in crushing is carried off by two Challenger ventilators, diameter 3·94 feet (1·2 metres), width 0·92 feet (0·28 metre) which is capable of carrying off 1,400 cubic feet (400 cubic metres) of dusty air when making 300 revolutions per minute.

The dust is collected in a suitable chamber. The separation of the blende from the iron is carried out by four magnetic machines, which are supplied with electricity by two dynamos, giving 50 ampères and 31 volts. The magnetic separating-machine consists of a revolving brass drum, inside which, and at one side of the drum, are fixed magnets. The ore is fed continuously at the outside of the revolving drum where the magnets are, and the iron is attracted by the magnets and falls into a compartment situated farther from the feed than that into which the blende falls. Twenty ampères is the maximum current allowed to each separating-machine. Each machine treats $1\frac{1}{2}$ tons of coarse ore and 1 ton of the fine ore per hour.

The blende is sometimes passed a second time through a magnetic separating machine, and is finally re-jigged to get rid of any gangue (slate, quartz amphibole, etc.) The cost of working is as follows:—

	Per Ton Treated. s.	d.	Per Ton of Product. s.	d.
Roasting	0	7·6	1	9·8
Crushing and sorting	0	6·0	1	5·0
Separation (magnetic)	0	6·7	1	7·2
Jigging	0	7·2	1	8·0
Total	2	3·5	6	6·0

The following tables show the amount of ore treated, and the product in 1891:—

ORE TREATED.

	Tons.	Percentage o Zinc.
1. Smalls (*grubenklein*)	2968·2	25
2. Rough ore (*erzwande*)	4576·1	24
3. Blende ore (*blende mittelerze*) ...	3223·8	27

PRODUCT.

Class of Ore.	Picked Ore.			Blende Concentrates.			Lead Concentrates.			Blende Ore.			Lead Ore.		
	Tons.	Zn.	Extraction.	Tons.	Zn.	Extraction.	Tons.	Pb.	Extraction.	Tons.	Zn.	Extraction.	Tons.	Pb.	Extraction.
		%			%			%			%			%	
1. Smalls ..	55·1	48·5	1·9	956·1	40·0	32·2	2·3	68	0·08	838·0	27	28·2	82·6	8	2·8
2. Rough ore	74·2	46·5	1·7	—	—	—	—	—	—	3,476	27	75·9	169·4	7	3·7
3. Blende ore	—	—	—	1246·0	43·5	38·6	2·4	68	0·07	—	—	—	—	—	—

W. F. W.

THE CONKLING MAGNETIC ORE-CONCENTRATOR.

Ore-Dressing by Electricity at the Tilly Foster Mine. *By* F. H. McDowell *Transactions of the American Institute of Mining Engineers*, 1890, *vol. xix. pages* 71-73.

The results of a six months' trial of the Conkling electric-magnetic separator are given. The local difficulties were great. Only about two-thirds of the waste-heap is ore, and this varies in content of iron from 20 to 28 per cent. In addition, the heap was originally located without reference to further treatment, and now can only be handled by means of a locomotive operating over heavy grades. The mineral is so widely and finely disseminated through the ore that it has been necessary to resort to fine-crushing.

The process includes crushing the ore by a Blake rock-breaker on the heap, removing it in train-loads to the bins in the mill, passing it under two ball-stamps provided with screens of $\frac{5}{16}$ inch mesh, elevating it to the Conkling electric separating belts, and delivering the concentrates to the wagons and the tailings to the settling-reservoirs.

In six months there were treated 18,058 tons of lean ore, giving an average of 3,009 tons per month; this produced concentrates amounting to 6,236 tons, so that 2·89 tons of crude ore were required to produce 1 ton of concentrates. The average cost of treatment, including preparing the ore and getting it to the mill and disposing of the tailings (which items make up one-half the labour charge) was 9·375 shillings per ton of concentrates. The actual expenses in the mill averaged 6·75 shillings per ton of concentrates.

Several conclusions are drawn from the results obtained, the chief one being: Unless the location and other conditions are exceptionally favourable, it will not pay to erect works to treat the material of waste-heaps carrying less than 25 per cent. of iron. H. W. H.

MAGNETIC CONCENTRATION OF IRON ORE.

By Harvey S. Chase. *Technology Quarterly* (*Boston*), *vol. v., pages* 64-69.

The paper deals with the separation of low-grade magnetic oxide, as found in great veins throughout the Appalachian range. The ore is reduced by repeated crushing, usually by rolls, to a fineness which generally varies between 4 and 16 mesh. Finer crushing, even up to 50 mesh, is sometimes necessary; but although an excellent product is often obtained in such cases, it is doubtful to what extent ores which require such fine crushing can be made to pay. The author adopts the plan of a preliminary coarse crushing and separation into heads for shipment, tailings, which are thrown away, and one or more grades of middlings for subsequent re-crushing and re-concentration.

The Chase magnetic ore-concentrator, consists essentially of a spirally-wound magnetic wheel, around which the crushed ore is carried by a canvas belt; worthless tailings are thrown off by centrifugal force, and the remainder is carried round to the under side of the wheel. Thence the belt passes under another magnet, with poles of varying strength and distance, by which the particles are made to rotate and tumble around each other. In this stage the middlings are separated into the number of classes desired, and the heads are carried along the belt to another smaller magnetic wheel, where the dust is removed by a current of air or water, and the ore passes thence on to the bin. The machine works either dry, or wet in a water-tight tank, the former giving quicker, the latter better, results. Some ores are best adapted for working dry, and others wet. G. E. C.

THE TREATMENT OF TAILINGS BY THE LÜHRIG SYSTEM.

Description of the New Concentrating Plant of the Himmelfahrt Mine, near Freiberg in Saxony. *By* O. BILHARZ. *Special Report of the Department of Mines, Victoria*, 1892, 18 *pages*, 3 *plates.*

The Lührig concentrating plant at Himmelfahrt, near Freiberg, Saxony, which has been in operation since 1889, is arranged in two equal and distinct portions, so as to admit of the separate treatment of ores from different parts of the mine, or of dressing ores from other mines. Its full capacity is about 140 tons per day, the average output from the Himmelfahrt mine being rather above this figure; but about one-fifth of the ore is separately treated by dry stamping and hand-picking, so that the average quantity of ore passing through the concentrating plant is about 118 tons per day.

The ore, which is first roughly separated by hand, consists chiefly of galena, iron and arsenical pyrites and blende, in a gangue of gneiss.

The galena contains generally from 49 to 55½ ounces of silver per ton, and as a rule is richest when fine-grained. The iron pyrites and arsenical pyrites contain no silver; the latter is separated as an ore of arsenic. The blende contains a high percentage of iron; and its specific gravity is very nearly that of the iron pyrites. Separation, therefore, even with the best classification and jigging, is very difficult; and the blende ores are, as far as possible, separated by hand and treated by themselves.

The main principles, to carry out which the plant was designed, are :—

(1) Disintegration of the ore step by step, with concentration at each stage.

(2) Effecting crushing and concentration as far as possible with rolls and jigs, rather than stamps and slime apparatus, so as to avoid the greater loss in the latter.

(3) Automatic working throughout.

It is hardly necessary to point out that these principles are in no way novel.

The situation chosen not giving slope enough for the latter purpose, it was necessary to erect the main building in several storeys, the ore being raised by a lift from the tramway leading from the mine.

The main building, constructed for the roll and jigging work, consists of four working-floors. The ore, in trucks containing about 2¼ tons each, is raised by the lift to the top floor, and there tipped into hoppers according to its quality. Thence it passes over a sort of shaking sieve which lets the smalls through and feeds the larger pieces into stone-breakers, whence, together with the above smalls they go on to sieve-drums and, after sizing, to jigs. The outfall from these sieves falls upon moving inclined picking-bands, where the pure ore is picked out, and the dradge passes on. This then goes through three sets of rolls, each with its arrangement of sieves, jigs, etc., the whole process being nearly automatic. The waste from the jigs is run by means of a launder to a building outside, and is sold as building-sand, gravel, etc.,

The mixed dradge resulting from the above processes is elevated into hoppers feeding 2 sets of American stamps, of 15 heads each, also situated on the lowest floor of the main building. Each stamp weighs 313 lbs., and makes 53 falls of a trifle over 1 foot per minute. The duty is 176 lbs. per head per hour, a total of 23½ tons per day, or about 20 per cent. of the whole quantity of ore treated by the plant.

The stamped ore is passed through sieve drums, then through *spitzlutten*, after classification being jigged so as to obtain the separate ores in a concentrated form.

In the final working the stuff is again passed through two more sets of *spitz-*

lutten. The first set divides it into coarse, middle, and fine sand. The coarse is treated with jigs of the usual kind; the middle and fine are first concentrated in a circular pulsating-jig divided radially into compartments, and which can treat the whole of the fine sands from the plant. Finally they are passed over percussion tables or vanners with rubber band moving 13 to 14 feet and receiving about 140 percussions per minute. The second set gives very fine sand and slime; the former is concentrated by a single-sieve percussion-jig, the latter by double *spitzlutten*, and both are then passed over similar percussion-tables.

The total number of men employed in and about the whole plant is 51, besides overseers. The steam-engine working the plant is of about 94 horse-power effective; and gives power for the lift, lighting by electricity, etc.

The quantity of water required is 35½ cubic feet per minute.

The tailings contain silver and lead per ton as follows:—

	Silver. oz.	dwt.	gr.		Lead. oz.
From the coarsely-crushed grains	1	12	16	...	nil.
„ elementary jigs	0	6	12	...	„
„ round jigger	0	13	1	...	„
„ percussion-tables	0	19	14	...	326¾
Deposits in settling tanks	3	5	8	...	653¼

No particulars are given of the cost of erection.

In another appendix to the departmental report are given the results of various trials made with Lührig vanners on various small parcels of tailings from Victorian mines. In the report Mr. Newbery emphasizes the fact that with the Lührig system success cannot be obtained if the supply of water is insufficient; and states that Mr. Lührig prefers the old *spitzkasten* to any other form of classifier.

G. E. C.

MAROS WASHING-TABLE.

Table à laver du Maros. Traitement des roches pauvres pour pyrites riches. By — Effère. *Génie Civil*, 1892, *vol. xxii., pages* 92 *and* 93.

The table here described is merely a modification of the ordinary hand-dressing box often used in Hungary for concentrating auriferous pyrites, and which is similar to the grave buddle once used in Cornwall for dressing tin. It consists simply of a sloping box, at the upper end of which the heavy sand, resulting from a rough gravity-separation of the pulp from the stamps, is placed, while a gentle flow of water washes it down, the denser particles being deposited, an attendant keeping it in motion. The Maros table consists of two such boxes placed side by side, successive streams of pulp and clean water being made to flow over them alternately, so that one attendant serves for both, and the dressed product can be washed off as soon as a thin coating is deposited. G. E. C.

RIGAUD CRADLE FOR WASHING ALLUVIALS.

Appareil de lavage des sables métallifères.—Berceau Rigaud. By — Effère. *Génie Civil*, 1892, *vol. xxii., pages* 120–122.

This is a form of cradle, without rockers, suspended from a raised longitudinal axis. The principal novelty appears to be the use of a kind of comb, fixed to the axis, which when the cradle is oscillated agitates the surface of the sand collected between each pair of riffles. Small holes are made in front of each riffle, through which the concentrated product can be withdrawn at intervals into a receptacle beneath G. E. C.

CASTELNAU SYSTEM OF ORE-DRESSING.

Laverie des mines de plomb argentifère de Bouillac (Aveyron). *By* F. DESQUIENS. *Génie Civil*, 1892, *vol. xxii., pages* 202-203, *plates XIII. and XIV.*

The principal features of this installation are the circular jigs and belt slime tables on the Castelnau system; which are, however, not described in much detail, and of which no drawings are given.

The ore, after passing twice through stone-breakers and over hand-picking belts, is crushed by two pairs of rolls, of 27½ inches diameter, and with a tangential velocity of 2·6 feet per second. It is then sized by a series of 5 trommels with holes of from ½ to 7 millimetres. The particles rejected by the first are re-crushed, and these passing through the last go to the slime-tables. The intermediate sizes go to the jigs, as to which the following figures are given:—

Diameter of Particles. Inches.	Millimetres.	Blows per Minute of Piston.	Length of Stroke of Piston. Inches.	Millimetres.
0·20 to 0·28	5 to 7	240	1·18	30
0·12 „ 0·20	3 „ 5	260	0·79	20
0·06 „ 0·12	1½ „ 3	280	0·40	10
0·02 „ 0·06	½ „ 1½	300	0·20	5

The product averages 60 per cent., and the waste 0·5 per cent. of lead. The middle product, of 10 to 20 per cent., is sent back to the rolls.

The slimes, after settling and re-mixing to the proper consistency, are concentrated on rubber endless belts with a surface of 46 by 4 feet, moving longitudinally 9·8 inches per second, and inclined in the direction of the width. Each machine is stated to concentrate 20 to 30 cwts. of ore per hour.

All the water is used over and over again, in order to avoid loss in floating slimes.

No particulars are given as to capacity, etc., of the plant. G. E. C.

RECENT GOLD-MILLING PRACTICE IN NOVA SCOTIA.

By JOHN E. HARDMAN. *Journal of the Mining Society of Nova Scotia*, 1892, *vol. i., part* 2, *page* 34, *and Canadian Mining and Mechanical Review*, 1892, *vol. xi., pages* 134-137, *and three plates.*

The Oldham mill has 10 stamps, especially designed for hard quartz containing coarse gold, and for the conditions of custom-milling. The battery is worked by power from a Pelton wheel.

The main features of the system of working adopted are a very short rapid drop and saving of most of the gold inside the mortar. The stamps are of the Homestake pattern, full stamp with new shoe weighing 858 lbs. The modifications adopted in the arrangement are fully described in the paper.

The stamps work 85 drops of 6½ inches per minute, the screen being the equivalent of a 30-mesh. The capacity is from 28 to 30 tons per 24 hours.

The gold is obtained in the following proportions:—On outside plates, 8·55 per cent.; on inside plates, 23·40 per cent.; and in mortar sands, 68·05 per cent.; so that 91·45 per cent. is retained behind the screen.

The tailings assayed 8s. 4d. per ton, almost entirely in the sulphides; tailings from concentrators gave 1s. 8d. to 2s. 1d. per ton.

The milling cost in the Black Hills is, according to Prof. Hofman, allowing deductions for fuel, 2s. 4d. per ton.

The costs per ton at the Oldham mill are as follows:—

	s.	d.
Labour	1	1·65
Supplies	0	1·47
Iron	0	1·32
Quicksilver	0	1·02
Total ...	1	5·46

When running on large lots the cost has been reduced to 1s. 2·31d. per ton. On the other hand, during the winter months an extra charge of 1·76d. is necessary for fuel to warm the mill. G. E. C.

MINING IN SARDINIA.

Histoire de l'Industrie Minière en Sardaigne. *By* — DE LAUNAY. *Annales des Mines*, 1892, *series* 9, *vol. i., pages* 511-538.

Mining was probably introduced into Sardinia by the Carthaginians. In Roman times, and up to about 1860 only argentiferous galena was worked to any great extent, the silver being extracted by cupellation. The galena worked by the Romans seems to have been only in soft rocks, such as limestone, and was mined by means of very numerous narrow vertical pits. Analyses of slags and pigs of lead found prove that the silver-extraction was well performed: the slags resulting from the initial production of raw lead, however, assay as high as 25 to 30 per cent. of lead, with 70 to 100 grammes of silver to the 100 kilogrammes of lead (23 to 32½ ounces per ton).

After the retirement of the Romans, the mines remained unworked until the Pisan domination (1004-1323), when they were worked by free companies of miners. The author mentions a curious mining code relating to this period, the technical terms showing traces of German influence. In some cases the workings were carried to a depth of 800 feet (250 metres), the drainage being performed by primitive pumps. The slags show an improved metallurgical treatment compared with that of the Romans.

Early in the fourteenth century Sardinia was conquered by the Aragonese, by whom the mines were worked as State property—at a loss: subsequently they were leased to a Genoese company. From the sixteenth to the nineteenth century mining was almost extinct. Its revival resulted from the application in 1848 of a new mining law similar to the French law of 1810, and a subsequent improvement of the same in 1859. The following figures are given of the extraction of argentiferous galena:—

	Tons.
1846	17
1858	6,637
1860	15,228
1865-6	26,229

Calamine was first worked in Sardinia in 1866 at the now famous Malfidano mines, where its value was discovered accidentally after a great deal of time and money had been spent fruitlessly in the search for galena. The mines now employ 1,700 men, and produce 50,000 tons of calamine annually. Up to 1888 the total production was over a million tons.

The first lead mine re-worked in modern times was that of Monteponi. Work was commenced in 1791 and carried on by the State on a small scale. In 1850, it

was leased privately for 30 years; and shortly after yielded an annual production of two million francs, of which 20 per cent. was profit. At the renewal of the concession in 1880, much more severe terms were exacted, including the driving of a drainage-tunnel 6,600 yards (6 kilometres) long; which was completed in 1890 at a cost of £80,000 (2,000,000 francs).

The Sarrabus silver mine was first re-worked in 1870, and in 1885 yielded a value of £96,000 (2,400,000 francs). The lode is extensive and well-defined, the silver being mainly in the form of sulphide, in a calcitic gangue.

Of late years the activity of mining in Sardinia has diminished, the production both of calamine and blende showing a falling off; this is owing partly to the increased costs of working in depth, partly to the fact that the calamine-deposits are always limited masses localized near the surface. Moreover, at the Sarrabus, it seems that below a certain depth the silver disappears, at any rate temporarily.

G. E. C.

MINING IN SARDINIA.

Geognostische und bergmännische Mittheilungen über den Bergbaubezirk von Iglesias auf der Insel Sardinien. *By* — MARX. *Zeitschrift für das Berg-, Hütten-und Salinen-Wesen im Preussischen Staate*, 1892, *vol. xl., pages* 263-278.

This paper contains a description of the geological features, and of the mines of the district of Iglesias in the south-western part of the island. The formation of the district is mainly Palæozoic, and these rocks rest upon granite, which is found at the surface in only a few places. Ores are found chiefly in the granite and in the slate formation, but also in limestones of more recent age.

In the granite the ore occurs in lodes, and the principal minerals found are silver-lead, zinc blende, fahlerz, and galmei. The vein-stuff is mostly quartz.

The limestone ore-deposits consist of lead and zinc ores, and occur as stockwerks or as contact-deposits between slate and limestone. In the granite district the most important mine is the Monte Vecchio, situated on the vein of that name, which is a contact-deposit between slate and granite. The principal minerals won are galena, zinc blende, siderite, and copper pyrites. Of the limestone mines the chief is the Monteponi mine, situated close to the town of Iglesias. In 1889—6,000 tons of lead ore were raised from this mine; it formerly belonged to the Italian Government who sold it to the present company in 1880. One of the conditions of sale was that an adit should be driven to drain the mines of the district. This adit, which commences 9 feet above sea-level, is now about 17,000 yards (15,786 metres) long. The ores are brought down to the coast by a railway 9 miles (15 kilometres) in length.

At the Monteponi mine the zinc ores, which contain on an average 35 per cent. of zinc, are roasted in a reverberatory furnace, and in an Oxland calciner. Cardiff coal is used. The reverberatory furnace treats 20 tons of ore in 24 hours, using 15·11 per cent. of coal. The ore loses in roasting 23 per cent. in weight. The Oxland furnace roasts 12 tons of ore in 24 hours, using 12·41 per cent. of coal. The ore loses 21 per cent. in weight. The reverberatory requires five men to attend to it, and the working expenses per ton of zinc ore amount to 4·20 lire (3s.) The Oxland calcining kiln is worked by two men, and 1 ton of roasted ore costs 5·40 lire (4s. 3d.)

The paper also contains particulars of the methods of mining. W. F. W.

ORE-MINING IN SERVIA.

Der Erzbergbau in Serbien. *By* F. B. PFEIFFER. *Berg-und Huettenmænnische Zeitung*, 1892, *vol. li., pages* 2-5.

The mineral resources of Servia, though as yet only partially developed, are very great, metallic ores of considerable variety occurring in various geological formations and in nearly all the high lands of the country. According to position they belong chiefly to the five principal districts of Kopaonik, Schumadija, Drina, Kucevo, and Zaplanina. So far as is known the Romans were the first to open mines in the country, and they were followed during the period of national independence by the Servians themselves. At present operations are conducted on a reduced scale in some only of the above-mentioned districts.

Gold is found in combination with various ores and in the sands of several rivers, and the ores of arsenic, antimony, bismuth, copper, iron, lead, manganese, silver, and of nickel and cobalt in small quantities are known to occur. Quicksilver is found in the district of Rudnik and in the Avala mountains. Silver occurs in the greatest quantities and in combination with lead forms the backbone of the mineral wealth in all the districts, the other metals occurring here and there in larger or smaller detached ore-deposits. After lead and silver, as regards quantity, come copper, antimony, and iron.

The Kopaonik mountains run from north to south for about 30 miles in the south of Servia near the Bosno-Turkish boundary. Now covered with woods and thinly inhabited, they are rich in ruins of towns, villages, and foundries. Heaps of slag and ore abound, and it is known that during the middle ages miners from Saxony and from Ragusa frequented the district in considerable numbers. The Turkish conquest put an end to the mining industry, and although probably far from exhausted, the district is now the one least worked and least known. It is known to contain iron, lead, silver, and gold.

The Schumadija district comprises the Rudnik mountains and their spurs. Its history corresponded with that of the Kopaonik district up to the date of the Turkish conquest, after which the Turks and subsequently the Austrians carried on mining work, chiefly in lead and silver. In the Avala mountains belonging to the Rudnik range slag from old workings is found, computed at about half a million tons. The chief veins run from Rudnik northwards, silver-lead ore being the most plentiful. Copper and gold occur in small quantities, though not very pure, while in the Vencac mountains pure iron ore occurs in large quantities along with the silver-lead ore. In the Avala range quicksilver is found, and there is a vein of iron ore reaching to the neighbourhood of Belgrade. Silver occurs in all the ores of this district, hitherto proved, in large quantities. As yet no lead ore has been found to contain less than 0·2 per cent. (65 ounces per ton) of silver, and some ores have yielded as much as 0·5 per cent. (163 ounces per ton). It has been estimated that the slag-heaps from old workings lying between Paryan and Guberevac contain 55 per cent. of lead and 0·0037 per cent. of silver weighing 28,000 tons and 37 tons respectively and worth over £1,000,000.

The Drina district is also rich in ores. From Loznica on the Drina to the Tara mountains and from Lesnica to Suvobor run veins of silver, lead, and gold, accompanied in places by iron, arsenic, zinc, and copper ores. From Borina to the neighbourhood of Valjevo runs a thick and rich vein of antimony which in several places crops out at the surface. This is accompanied on the left by zinc, lead, and silver, and on the right by copper, the latter vein reaching its greatest thickness near Valjevo where it comes to the surface. The silver-lead ores of this district contain from 0·25 per cent. (82 ounces per ton) to 0·55 per cent. (180 ounces per

ton) of silver. The antimony was not worked in earlier times. Zinc occurs in large irregular blocks containing from 8 to 50 per cent. of pure metal near the village of Vasic. The whole Drina district is traversed by paths, now overgrown, and the numerous old cemeteries bear testimony to the large population that once inhabited it. The two best known ancient mining centres of the district were Zajeca and Krupanj, the latter place being at one time the centre of the iron industry of all Servia.

The ore district of Kucevo comprises the mountains in Eastern Servia which form a continuation of the Siebenburg and South Hungarian Carpathian mountains, through which the Danube has broken the passage of the Iron Gate. The same formations occur on both sides of the river. The principal mines now at work are the Kucajna and Majdanpek. The Kucajna numbers lead, silver, and gold among its ores, the Majdanpek lead, iron, copper, silver, and gold. The Kucajna ores are the richest in gold and silver in all Servia, containing, as they do, 0·07 per cent. (22 ounces per ton) of gold and 0·75 per cent. (245 ounces per ton) of silver. This district was also worked by the Romans, as by later masters of the country in succession.

The Zaplanina district contains the mountains between the Nisava and the Binacka-Moruva. Its mineralogical features are now little known, but there are many traces of the existence in earlier times of a busy mining industry. Veins containing iron, copper, and silver crop out in various places, and gold occurs in the rivers. The river-beds were searched for gold in olden times; and even now, gipsies do a little gold-washing, although it is forbidden by law.

A. R. L. and W. F. W.

MINING AND METALLURGY IN CHILI.

Le Chili Minier, Métallurgique, Industriel. *By* CH. VATTIER. *Mémoires de la Société des Ingénieurs Civils*, 1892, *vol. ii, pages* 37-140, *and plate LXVII.*

The raw nitrate earth as extracted in the Tarapacá and Taltal districts, varies in contents from 20 to 40 per cent. of pure nitrate. The average cost of extraction is about 25s. per ton (70 centavos per quintal of 100 lbs.) With raw nitrate of 50 per cent. 1 quintal of coal is required to produce 12 of pure nitrate. The cost for rail carriage (66 miles) is 13s. per ton (37 centavos per quintal).

Gold is found both in alluvials and lodes. Among the latter are cited those of Guanaco in Taltal, where the veinstone consists of quartz and barytes, often crystalline and transparent, with spangles of gold. In the province of Coquimbo many lodes are worked, the yield rarely exceeding 11-14 dwts. per ton.

The principal silver-mines worked are situated in the north, especially in the provinces of Antofagasta and Atacama. Those of Copiapo, which produced largely some forty years since, are now mostly worked out. In the province of Santiago are situated the well-known mines of Las Condes.

The copper-mines of northern and central Chili constituted for a long time the principal mineral riches of the country. The production nearly forty years ago was one-third of that of the whole world, whereas it is now less than one-fifteenth. At first only the oxidized ores were worked, the sulphides being rejected; during the last half century, however, the sulphide ores having given the bulk of the output. The principal centre of production is the province of Coquimbo.

Chili is exceptionally rich in deposits of iron ore of high quality, often situated under conditions very favourable for working and transport; they are especially abundant in the provinces of Atacama and Coquimbo. As yet they have only been

worked to a comparatively small extent for fluxes for smelting other ores. The average contents in metallic iron are given as 66·05 per cent. Very little attention has been given to iron in the south, but deposits are known to exist in the vicinity of Coronel and Valdivia.

The most important manganese deposits are situated in the neighbourhood of Carrizal Bajo and Coquimbo. A typical sample contains peroxide of manganese 56 per cent., protoxide 24 per cent., silica 5 per cent., carbonate of lime 9 per cent., peroxide of iron 0·60 per cent., and phosphorus 0·03 per cent. As a rule, both the quality and percentage of the ore deteriorate at a small depth, and in some cases it disappears altogether at a depth of 14 or 15 yards.

The great bulk of the copper ore is treated by smelting, either in reverberatory furnaces or *fours à manche*. Wood is used for smelting only in Coquimbo and a few of the departments in the centre. The cost in Coquimbo, delivered to the furnace, is about 5s. to 6s. (3 to 4 pesos) per ton. At Maitenes in Santiago, the dry wood used for cupellation, etc., costs as much as 21s. to 22s. (13 to 14 pesos) per ton. In the south excellent wood can be procured at less than 3s. (2 pesos) per ton. Charcoal has not been used hitherto for smelting, but an excellent product could be made in the south and delivered at 16s. (10 pesos) per ton.

The Chilian lignites are produced mainly near Concepcion: other deposits exist farther south, but have not yet been worked. The present production is 600,000 tons per annum, and is rapidly increasing: the price per ton is now about 16s. to 19s. (10 to 12 pesos) or more, delivered at the northern ports. The beds are near the coast, and vary in thickness from 2 to 5 feet. They are often worked for considerable distances under the sea: gas is sometimes developed, but accidents are rare.

The following are analyses of lignite from Lota:—

Water	5·0	...	4·8
Volatile matter	40·2	...	40·8
Fixed carbon	53·2	...	48·2
Clay and ferruginous ash	1·6	...	6·2
	100·0		100·0

Calorific power as compared with pure carbon, 75·5 and 71·3. Foreign coal, being brought as return freight at low rates, costs only 3s. to 5s. (2 to 3 pesos) more per ton than this lignite. The foreign coke used for smelting costs from 46s. to 52s. (29 to 33 pesos) per ton, and anthracite from 27s. to 33s. (17 to 21 pesos) per ton delivered.

Silver ores are in Chili treated either by amalgamation or by smelting in low blast-furnaces. The former is now performed by a method known as the Kronke process, the old patio process having long been abandoned. This process appears to consist of the addition of chemicals, notably salt and sub-chloride of copper, during amalgamation.

The small amount of lead in the ores, and their siliceous nature, necessitate many special features in the smelting process adopted, which is described in great detail. In the smelting mixture a great deal of slag and fluxes are necessary, generally nearly half the total charge exclusive of coke; the coke consumed is 25 per cent. or more of the actual ore smelted. The ore must be in lumps: fines or dust must be made into briquettes, and even then used sparingly. The average quantity smelted will be, with ordinary ores, 10 to 12 tons per day. The product is a copper-lead matte, containing generally about 30 to 35 per cent. of copper, 16 to 18 per cent. of lead, and 0·1 to 0·3 per cent. of silver. Some argentiferous lead is produced at the same time; which, after re-smelting, contains 90 to 92 per cent. of lead, with 0·8 to 1 per cent. of silver.

The average assay of copper ores, as sold to the smelting-works, is about 12 to 14 per cent., but in special cases it is much higher or lower. In some parts of Chili, in the centre and south especially, there are many small reverberatory furnaces worked by the landed proprietors, using wood as fuel, and producing a 50 per cent. matte at one operation. This is broken and roasted in heaps, and when sufficient is collected it is re-smelted to bar copper in the same or a similar furnace. These furnaces smelt 3 or 4 tons per 24 hours, using 10 tons of wood costing 3s. to 5s. (2 to 3 pesos) per ton: and, in consequence of the growing scarcity of wood, they tend to disappear.

The greater part of the Chilian copper is however smelted in reverberatory furnaces with foreign (English or Australian) coal or native lignite, by the ordinary Welsh process, producing 50 per cent. matte. This is broken, crushed, roasted, and re-smelted with oxide ores and slags, yielding some copper bars and some white matte of 70 to 72 per cent. The latter is re-roasted, made into rosette copper and refined. The 50 per cent. matte is, at Lota, Bessemerized direct.

In these reverberatory furnaces about 1 ton of coal is necessary to smelt 2 or 3 of ore; and the slags from the first fusion assay 0·75 per cent. on the average. Many of these works are situated in the province of Coquimbo, where the Guayacan works have produced as much as 1,000 tons of copper per month. Still more are found farther south in the coal region, where the ores are mainly imported from the north. The Lota works have produced up to 12,000 tons per annum of copper bars.

A good deal of ore, especially away from the coast, is smelted with coke or anthracite, in low blast-furnaces (*fours à manche*) much resembling those used for silver ores. The method of smelting at the Maitenes works, province of Santiago, is described. The ore averages 18 to 25 per cent.; it is a double sulphide of copper and iron, and the gangue is quartzose. It is screened, and the coarse portion roasted in heaps of 200 to 300 tons. The fines are made into briquettes and roasted, but this operation at present is difficult. The coke used is 13 per cent. of the ore weight. Each furnace smelts from 20 to 25 tons of ore (besides fluxes, etc.) per 24 hours. The slag assays under ½ per cent., and the resulting matte about 50 per cent.; it is led directly into a converter, where it is transformed into copper bars. Each converter produces 4 to 5 tons of copper per 24 hours.

At the works of the Panulcillo Company 308 tons per month are produced of 50 per cent. matte, from a very poor ore with a garnet-rock gangue.

The author strongly advocates the creation in Chili of a steel-making industry, for which he says every facility exists, either by using coke made from the lignite (if the latter will admit of it) or charcoal from the southern forests.

G. E. C.

PROGRESS OF THE METALLURGY OF NICKEL.

Les Progrès de la Métallurgie du Nickel, etc. *By* D. Levat. *Annales des Mines*, 1892, *series* 9, *vol. i., pages* 141-226.

The only nickel-ore deposits of known first-rate importance are those of New Caledonia and Sudbury in Canada. In New Caledonia, nickel occurs only in the form of hydrated magnesian silicate, apple-green in colour when pure, in fissures in serpentine; and it must clearly have been originally deposited in this state, sulphides or arsenides being unknown, even in the deepest workings. The serpentine forms about half the area of the island, especially towards the south: the nickel being always found at or near its junction with funnel-shaped pipes (*vasques*) of red clay, although never in the clay itself.

This clay is formed by the hydrothermal alteration of the serpentine; numerous fissures in which, bearing north-west and south-east, or roughly at right-angles to

the general direction of the island, have given passage to metalliferous springs carrying iron and manganese, which have eaten away the serpentine, and deposited their dissolved matter in the cavities so produced. The manganese, always containing cobalt, is found as beds in the clay: and the iron solutions having finally predominated, great masses of granular hydrated hæmatite have been deposited on the top of the clay. The cobalt is in the form of hydrated oxide, without any trace of sulphur or arsenic, and the mineral extracted rarely contains more than 2·5 to 3 per cent. of metallic cobalt.

In the south of the island the serpentine contains many grains of chrome iron, which have been re-arranged in an enriched form in the clay.

After the formation of the clay-pockets, fissures and cavities have been produced between the clay and the surrounding serpentine; which, together with the fissures and joints in the serpentine itself, have been subsequently filled with the deposits from nickeliferous solutions, forming a breccia, which constitutes the nickel ore. The deposits are therefore either stockworks, contact deposits between the clay and serpentine, or the fillings of fissures in the serpentine near the clay: these last being generally more concentrated and therefore more valuable.

The stockworks are worked as open-cuttings; and it is very important in commencing to entirely remove the red clay without allowing it to mix with the nickel ore—first, because the grains of iron-ore which it contains cannot be separated easily by washing from the nickel, and much impoverish the matte: and, secondly, because the highly aluminous clay renders the mineral, already very siliceous, still more refractory. This mode of working is very troublesome during the rainy season, particularly as the labour supply is small.

The ore is divided into two classes—that containing over 8 per cent. of nickel, and poorer ore. It is then brought down to the plains for washing, which consists in merely separating the red mud; owing to the equal density of the ore and gangue, no further concentration is possible, and all ore of 3 to 4 per cent. and under has to be thrown away.

The mineral as extracted varies greatly in composition, but the following may be taken as extreme analyses:—

	Per Cent.		Per Cent.
Silica	45	...	50
Iron	16	...	14
Nickel	8	...	7
Magnesia	12	...	10
Alumina	3	...	5
Water and oxygen ...	16	...	14
	100	...	100

This composition necessitates an addition of 25 to 30 per cent. of bases, and more or less sulphides, the smelting mixture being made up as follows:—

	Cwts.
Ore	20·0
Coal	6·0
Sulphur	0·7
Fine coke	1·5

The high cost of coke, and the want of experienced labour for smelting, has caused the concentration of the ore to a matte on the spot to be temporarily suspended. It is now principally treated in England in small water-jacket furnaces running through 25 to 30 tons of ore-mixture per 24 hours, soda-ash being used as a flux. The coke used is over 30 per cent. by weight of the ore, and the resulting matte averages 50 to 55 per cent. of nickel, 25 to 30 per cent. of iron, and 16 to 18 per cent. of sulphur.

At Sudbury, in Ontario, Canada, nickel is found in association with magnetic iron pyrites, accompanied by chalcopyrite—the mines having been first worked for copper only. The mineral occurs in large lenticular masses, interstratified with gneiss of the primitive formation of North America—here over 3,000 feet thick. The beds have been greatly contorted, lying at steep angles up to 70 degs., with a general strike of north-east and south-west: and they are frequently penetrated by dykes of diorite, the deposits of magnetic pyrites occurring at or near the contact, sometimes even in the diorite itself—which forms, moreover, the gangue of the ore.

The average assay of the ore rarely exceeds 3 to 4 per cent. of nickel, and about the same of copper, but it appears to become enriched in depth. The following is an average analysis of the ore picked out for copper:—

Sulphur	26·72
Copper	12·61
Iron	29·22
Nickel	3·12
Protoxide of iron	6·22
Lime	4·84
Magnesia	2·61
Alumina	2·63
Silica	13·06
	101·03

According to Messrs. Peters and Garnier, the deposits may be divided into two classes:—(*a*) Those such as the Stobie and Leighton mines, composed of almost pure massive magnetic pyrites containing little copper or nickel, but which after roasting form a good flux for the richer ores. These deposits are worked by open-cutting. (*b*) The deposits of more elongated lenticular form, worked by means of shafts following the dip. On an average the ores from these deposits (Copper Cliff mine, Evans mine, Blizzard mine, etc.) contain 3 to 5 per cent. of nickel and as much copper. All these ores are trammed direct to the roasting-floors, the cost delivered being about 8s. (10 francs) per ton, with about 50 per cent. of ore of the first class. The roasting is done in heaps, costing about 2s. 6d. (3 francs) per ton; a heap of 500 or 600 tons taking 50 or 60 days to roast. The roasted ore is smelted in large water-jacket furnaces, with a coke consumption of 12½ per cent. of the weight of ore; the matte produced being about 12 per cent. of the charge, and the slags assaying 0·45 per cent. of nickel and 0·40 per cent. of copper. Originally the matte was considerably richer in copper than in nickel, which greatly lessened its value: in 1889 the matte of the Canadian Copper Company ran 26·9 per cent. of copper and 14·1 per cent. of nickel; but in February 1891 this had been reduced to 16·9 per cent of copper against 21·5 per cent. of nickel. The matte produced by the Dominion Mineral Company, who are able to mix their ore with purely nickeliferous mineral from other mines, averages 18 to 20 per cent. of copper to 24 to 26 per cent. of nickel.

Coke costs about £1 8s. (35 francs) per ton delivered at the furnaces; and the average cost of smelting in the large water-jackets of 120 tons per 24 hours' capacity, may be taken at 6s. to 7s. (8 to 9 francs) per ton.

The paper includes a very complete account of the metallurgy of nickel; but the author merely refers to the considerable nickel-deposits of Norway, which occur under very similar conditions to those of Canada, and which have been largely worked during the last twenty-five years. G. E. C.

THE PRODUCTION OF NICKEL.

Om verdens nikkelproduktion og om konkurrance-betin-gelserne mellem de norske og de udenlandske nikkelforekomster. *By* J. H. L. Vogt. *Geologiska Föreningens, Förhandlingar, Stockholm,* 1892, *vol. xiv., pages* 433-475.

The Sudbury (Canada) deposits of nickeliferous pyrites are—as is also the case in Norway—situated in gabbro, and are often concentrated at its contact with other rocks; they are mineralogically and geologically similar to the Norwegian deposits. As in other cases, the nickel carries with it a little cobalt, generally about one part cobalt to five to ten nickel. The accompanying copper is also considerably more than in Norway and Sweden.

Nickel ores are of three kinds :—

Arsenides.—Found chiefly in lodes, and which up to the middle of the century formed the chief source of supply, from Germany and Austria-Hungary.

Sulphides.—As found in Canada, Norway, etc.

Silicates.—As worked in New Caledonia, etc.; and deposited by lateral secretion from serpentine containing a little nickel. In New Caledonia the garnierite occurs intimately associated with asbolite, and seems to have been derived from the same solutions; the latter being deposited first, probably because its component elements were easier to oxidize.

The author gives a table showing the world's production for the last half-century: rising from 100 to 250 tons yearly in 1840 to 1860, to 600 to 700 tons in 1870 to 1880, and 1,250 to 1,500 tons in 1888 and 1889. He estimates the production of 1890 at about 2,000 tons, and that of 1891 and 1892 still higher; but considers the figures given by Mr. Levat* to be excessive.

Details and tables are given of the output in the various producing countries, especially as to Norway, Germany, and New Caledonia.

He states the average contents of the Norwegian nickeliferous pyrites as 3 to 4½ per cent. of clean mineral. Of course the smelting-ore is much poorer, having averaged up to 1880 only about 1 per cent., and has since varied from 1·4 to 2·3 per cent. The garnierite mixture of New Caledonia averages of late years from 6 to 8 per cent.

There are also tables giving the fluctuations in price from 1867, when it was worth 8·10 kroner (say 9s.) per kilogramme, to the present value (at the date of the paper) of 3·50 to 4 kroner (say 4s. 5d.). Between 1873 and 1875 the price rose to 20 and 24 kroner (£1 1s. to £1 6s. 8d.). The rise which then reached its apex was owing to the demand for coinage purposes; the subsequent rapid fall being caused by the working of the New Caledonia deposits.

According to Mr. du Peloux the cost of production per kilogramme in New Caledonia of metallic nickel was (1880) 5s. to 6s. (6 to 7 francs), which could be reduced to 3s. or 4s. (4 to 5 francs). The author asserts that the cost in the best Norwegian works is 1·50 to 2 kroner (1s. 8d. to 2s. 3d.), including all costs for bringing up to a 50 per cent. matte; and gives Mr. Levat's authority for an estimate of the cost at Sudbury (Canada) as 2 francs per kilogramme in a 20 per cent. matte (not charging any of the cost to the copper contained in it). At this rate the costs of production would be nearly equal, the higher grade of the Canadian ore being counterbalanced by the higher prices for labour and materials. G. E. C.

* *Annales des Mines,* 1892, vol. i., page 141.

NICKEL MINES OF NEW CALEDONIA.

Étude sur les Mines de Nickel de la Nouvelle-Calédonie. *By* FÉLIX BENOIT. *Bulletin de la Société de l'Industrie Minérale*, 1892, *series* 3, *vol. vi., pages* 753-804.

An historical account is given of the discovery of nickel in New Caledonia by Mr. Jules Garnier, and of the subsequent development of the industry.

The deposits are found in the serpentine formation which constitutes the greater part of the south of the island. The mountain crests of this formation, on which they occur, run about north-west and south-east, or in the direction of the length of the island; their sides are covered by sheets of red clay surmounted by masses of iron ore.

The deposits consist entirely of hydrated magnesian silicate (garnierite), varying in colour from green to chocolate, soft and often unctuous to the touch. The composition is extremely variable, but the average is stated as follows:—

	Per Cent.
Nickel	7
Magnesia	26
Iron	12
Silica	45
Water...	10

The serpentine itself nearly always contains traces of nickel, said to vary from 1 to 5 per cent.

The nickel ore is always enclosed in serpentine, at or near its contact with the red clay. It occurs generally in very irregular fissures, often forming a kind of stockwork; but sometimes in definite almost vertical veins, or in masses. Many of the deposits are worked in the open. There is no gangue, as a rule.

The deposits are situated along definite lines of enrichment, starting from the east coast and running in a south-west direction; and at present only those near the coast are worked. The nickeliferous veins as a rule diminish in size in depth, but to this there are exceptions.

There is nothing worthy of note in the methods of working, either by levels or open cut. The mines are generally situated at a considerable elevation, averaging 1,500 feet, and the ore is transported from the principal mines, by inclined planes or aerial tramways.

The labour employed consists of the three following classes:—

(1) English miners, paid 6 to 9 francs (5s. to 7s.) per day.

(2) Kanakas, receiving 20 francs (16s.) per month, with keep.

(3) Convicts and other prisoners hired from the State.

The author strongly advocates the importation of free miners from France.

The cost of mining varies from 15 to 40 francs (12s. to 32s.), and that of carriage to port from 0·50 to 10 francs (5d. to 8s.) per ton. The value at port of shipment for ore of 7½ to 8½ per cent. nickel is 105 francs (£4 3s. 4d.), and ore of 9½ to 10½ per cent., 125 francs (£4 19s.) per ton; the assays being calculated on the ore when desiccated. The freight to Europe is from 40 to 50 francs (£1 12s. to £1 19s. 6d.).

Thio District.—The author next proceeds to describe the mines at work in great detail, commencing with those belonging to the French Company Le Nickel. This Company works eleven mines situated in the Thio district, which he divides into three groups; all are worked as open cuts, and they together employ 1,500 hands.

The first or Plateau group comprises five mines, together producing 2,500 tons monthly of ore averaging 7 per cent., a production which it is stated might at once

be raised to 4,000 tons per month if sufficient labour were available. One mine is worked to a depth of 300 feet. The whole mountain in which they are situated is said to be probably a stockwork.

The Pauline group comprises two mines, both of stockwork type, producing together 500 tons monthly of 5 to 6½ per cent. The Pauline mine is situated in greenish serpentine of porphyritic appearance, the alteration of which shows a definite line up to which the nickel-bearing solutions reached.

In the third or Stilling group the ore is friable, varying in colour from yellow to brick-red, so that it is difficult to distinguish it from the clay previously mentioned. The production of this group is 300 tons per month, averaging 7 per cent. nickel.

Nakéty District.—The Bienvenue mine is worked on a well-defined vein running north-west and south-east, and dipping regularly 80 degs. to the west, the workable thickness of which varies from 10 inches to 2½ feet. Both walls (serpentine) are well-defined, with slickensides. The nickeliferous filling varies in colour from green to chocolate-yellow, and black at the lowest level. This mine produces 300 tons per month of 8 per cent. ore.

The Barbouillée and Boulangère mines, both worked as open cuts, produce together 200 tons of 6 per cent. ore per month.

Canala District.—The workings of the Boakaine mine are on a mass of rich green ore apparently formed by the union of a number of small veins. It produces 250 tons per month of 8½ per cent. ore, and belongs to the Le Nickel Company.

Couaoua District.—The Dorée mine is worked by four levels; the ore being quartzose in the upper level, and magnesian in the lower ones. It produces 350 tons of 6 per cent. ore per month, which with additional labour could be easily increased.

The Caulry group are worked by quarries and mines on deposits, which in depth concentrate from stockworks into definite east-and-west lodes, coinciding in strike with the crest of the mountain in which they are situated. Their production is 300 tons per month, averaging 9 per cent.

The other principal districts are those of Bourail, Paouéa, and Kouiambo. It is estimated that the total production for 1892 will reach 100,000 tons. G. E. C.

PRODUCTION OF NICKEL IN THE UNITED STATES.

Nickel. By W. R. Ingalls. *The Engineering and Mining Journal (New York)*, 1892, *vol. liii., pages* 40-41.

Occurrence.—In the United States there is but one deposit of nickel which has been mined regularly for that metal alone, it being located at Lancaster Gap, Penna. The lead ores of South-eastern Missouri carry a small amount of nickel, in conjunction with cobalt, both of which are recovered as bye-products.

Production.—The following figures show the production of nickel in the United States, together with the imports into the country and the exports from it, for the years 1890 and 1891 :—

	Production.			Value of Imports.	Value of Exports.
	Amount. Lbs.	Average Price. s. d.	Value. £	£	£
1890 ...	200,332 ...	2 8 ...	26,694 ...	77,137 ...	97
1891 ...	144,841 ...	2 5¾ ...	17,816 ...	— ...	—

The exports of nickel from New Caledonia in 1890, according to a recent consular report, amounted to 3,300 lbs., and of nickel ore to about 5,000 tons, which, averaging about 8 per cent. of nickel, was equivalent to about 882,000 lbs. of metallic nickel, representing a total export of 885,300 lbs.

The production of nickel (in nickel-copper matte) in Canada in 1890 was 1,336,627 lbs., that being the first year for which statistics of the Sudbury district were published.

Methods of Nickel Winning.—The nickeliferous chalcopyrite and pyrrhotite of Sudbury, Ontario, are first roasted and then smelted in cupola-furnaces to nickel-copper matte, containing about 20 per cent. of nickel.

These mattes are very refractory, and had not been successfully handled until the year 1891. Of the large output in the Sudbury district in 1890 but a small portion was shipped. In the latter part of that year the Orford Copper Company devised a successful method of refining these mattes, and since then the bulk of the output of the Canadian mines has been brought to the United States, the Orford Copper Company now being the largest nickel refiner in the world. By the secret process used by this company, the copper and nickel in the matte are separated and the nickel converted into nickel oxide, which is said to be more suitable for the manufacture of nickel-steel than metallic nickel. A small part of the Sudbury mattes is sent to Swansea to be refined.

There are three nickel-smelting and refining works in the United States, viz., the American Nickel Works, at Camden, New Jersey, the works of the Orford Copper Company, at Constable's Hook, New Jersey, and the works of the Canadian Copper Company, near Cleveland, Ohio, which are not yet completed. The American Nickel works use ore from the Lancaster Gap mine, Canadian mattes, and ore from South-eastern Missouri. The Orford works run exclusively on Canadian mattes, whilst the Cleveland works are to use mattes from the Canadian Copper Company's mines at Sudbury, Ontario.

The Nickel Market.—The consumption of nickel has greatly increased, the increase being due principally to the growing demand for the metal for use in the manufacture of nickel steel, the Creusot works (in France) alone having contracted for a large part of the product of the *Société du Nickel*, while the United States Government purchased the large quantity of 6,500 tons of Canadian matte, containing probably about 20 per cent. of nickel. There has also been an increased demand from the nickel-platers, the German-silver manufacturers, etc.

The United States Government (1891) made elaborate tests of nickel-steel and other kinds of armour-plate at Indian Head, Maryland, which resulted in a decisive approval of the nickel-steel, and it has now been adopted as the protective material for the new cruisers and battle-ships. J. W.

THE HUANCHACA MINES, BOLIVIA.

Notes on the Huanchaca Mine, Bolivia, South America. *By* ROBERT PEELE, JUN. *School of Mines Quarterly* (*New York*), *vol. xiv.*, *pages* 152-155.

The mines are situated in a group of hills on the Central Bolivian plateau, 13,400 feet above sea-level, and are connected with the Chilian port of Antofagasta by a 30-inches gauge railroad, 395 miles long. The workings are on two parallel and nearly vertical veins, cutting through a mountain which reaches an elevation of 14,500 feet.

The mines were extensively worked some time ago to a depth of several hundred feet, with small success. During the last fifteen years, however, they have produced a value of over £9,000,000, of which about £3,000,000 has been paid in dividends. Until recently they were worked on Spanish-American lines, but during the last two years a large modern hoisting-and-drilling plant has been erected.

They are worked through a tunnel, which cuts the veins at 1,500 feet from its mouth. This has been extended through the mountain towards the reduction works, 4½ miles away, the remaining distance consisting of a gravity-road.

The lowest depth attained (1891) was 1,190 feet below the level of the main tunnel. During 1891 5,942,000 ounces of silver were produced, yielding a net profit of over £420,000. 17,193 tons of ore, containing 4,041,000 ounces silver were exported to Europe, and 13,890 tons, containing 1,894,800 ounces, treated locally. The cost of mining, handling, and sorting is about £5 15s., and that of milling £4 19s. per ton.

The gangue is quartzose, carrying iron and copper pyrites, galena, blende, and tetrahedrite. The latter carries most of the silver; it is usually massive, but sometimes beautifully crystallized.

The following is an average analysis of sorted ores:—

Silica...	23·00
Copper	1·89
Lead	11·30
Zinc	21·50
Iron	11·60
Sulphur	26·00
Arsenic	0·35
Antimony	1·80
Silver ...	from 120 to 300 ounces per ton.

The ore is pulverized in Gruson (ball) mills of German make, each mill grinding 10 to 12 tons per 24 hours to 50 mesh, 130 lbs. being crushed per horse-power per hour.

The ore is roasted in reverberatory furnaces with 4 per cent. of salt, 25 lbs. of brushwood being required per 100 lbs. of ore. After withdrawal from the furnace, the ore is spread out on the cooling-floors to prevent farther chlorination.

The total losses average:—

	Per Cent.
In transportation and grinding	1¼ to 2
In roasting	3 „ 5
In amalgamation	8 „ 10
In stolen amalgam	0·1
In melting into bars	0·6

The net extraction thus varies from 82 to 87 per cent., the silver averaging 994 fine.

New reduction-works are being built near the coast, and the low-grade ore—of which there is 60,000 to 70,000 tons, of 75 to 90 ounces per ton, in the dump alone—will in future be treated there, 395 miles away from the mine. G. E. C.

PHOSPHATES IN CANADA.

The Phosphate Deposits of the Ottawa District. By R. W. Ells. *Canadian Mining and Mechanical Review*, 1893, *vol. xii., pages* 39-40.

The deposits are of two kinds, the apatite, occurring in Laurentian rocks, and the phosphatic nodules, found in fossiliferous strata of Cambrian and Cambro-Silurian age. The latter have not been worked.

Apatite was first mined in Ontario some thirty years ago, but up to 1875 the production averaged only about 1,000 tons per annum. From 1878 to 1889 it was but little more—the output for these twelve years being only 16,000 tons. In Quebec, on the other hand, where mining commenced in 1871, a production was reached in 1885 of 28,535 tons.

It occurs in two ways—either in association with intrusive masses or dykes of pyroxene, cutting through the altered Laurentian gneisses, or as crystals, often of large size, disseminated through limestone. The more economically important deposits belong to the former class, the apatite being nearly always found in more or less connected pockets near the contact with the gneiss.

The origin of the deposits is briefly discussed—the conclusion being that they were formed by the action, on the calcite of the pyroxene masses, of vapours charged with phosphoric acid ascending along the line of contact, and impregnating the softened or heated mass in patches near the contact. G. E. C.

PHOSPHATES IN FLORIDA, UNITED STATES.

(1) *The Phosphate Industry of Florida.* *By* FLOYD B. WILSON. *Engineering Magazine* (*New York*), *vol. iv., pages* 80-94.

The phosphates of Florida may be divided into four classes—Peace river pebble, land pebble, hard rock, and mixed land pebble and plate rock. The Peace river pebble is pumped up from the river-bed, and, after drying, is ready for shipment. It is of about 60 per cent., and finds a ready market. The first discovery of these beds was made in 1889, and it is said by some that they are nearly worked out.

The land pebble is often of very high grade, but no great quantity has yet been shipped, difficulty being found in washing out the clay in which it is embedded.

The hard rock phosphate occurs in pockets of very varying size, there being usually an overburden of 10 or 15 feet above the beds. To remove impurities it has to be crushed, washed, and dried before shipment.

The mixed land pebble and plate rock is of very high grade, averaging from 77 to 79 per cent., and occurs in potholes on the jagged surface of waterworn Eocene limestone. These potholes are filled with small pebble and broken plate, and are sometimes as deep as 40 feet. There is a sand overburden varying from a few inches up to 3 feet in depth. The deposits extend only over an area of $2\frac{1}{2}$ by $1\frac{1}{2}$ miles. The phosphate is mixed with clay and sand, which are washed out, and it is then dried —the whole process being automatic. G. E. C.

(2) *Suggestions as to the Origin and Deposition of Florida Phosphates.* *By* WALTER B. M. DAVIDSON. *The Engineering and Mining Journal* (*New York*), 1891, *vol. li., pages* 628-629.

The phosphate field in Florida includes nearly the entire western half of the peninsula. The phosphates are divided by the author into—(*a*) hard rock or boulder phosphate; (*b*) soft rock; (*c*) nodular phosphate; (*d*) river pebble phosphate; (*e*) land pebble phosphate.

Until a few years ago, when Mr. Albertus Vogt and Colonel John Dunn found that there was phosphate in Florida, the whole peninsula was looked upon as a coral reef covered with a blanket of sand.

Now the interest is so universal and so much capital is at stake that the United States Geological Survey have ordered a reconnaissance; indeed, they are at present in the field, and their results will be looked forward to with interest.

Geologists here have a hard task, for there are practically no outcrops, no sections to be studied, no synclinals or anticlinals to be measured, and nearly the whole peninsula is covered with a mantle of sand. Many artesian wells have been sunk, the most valuable to geologists being that sunk at Lake Worth, on which Mr. N. H. Darton, U.S. Geological Survey, has published a monograph recently.

The author maintains positively, from his personal observations, that an Eocene or Miocene limestone underlies the greater part of Florida and, broadly speaking, the entire phosphate field. Referring to the wave or beach theory, advanced by Dr. Wyatt,* the author considers it very ingenious and distinctly scientific, and says that while, no doubt, wave action came into play, it does not explain satisfactorily the rock phosphate deposit in all its vagaries.

Apparently, during the close of the Cainozoic period the shores of the Gulf of Mexico were elevated by gradual upheaval until they formed shallow lagoons, estuaries, and bays. In these shallow, warm seas lived myriads of shell-fish, many secreting phosphate as well as carbonate of lime, as is shown by the analysis of a shell of *Lingula ovalis*, quoted by Dr. Dana, as containing 86 per cent. of phosphate of lime. Fishes of all kinds abounded in these waters, died, and their bones, while mostly disappearing, served to increase the amount of phosphate of lime in the limestone.

Gradually the shores emerged from the seas, and while they rose came the glacial epoch. The cold of this epoch drove all living creatures which could travel southward. The result was that the great mammal horde flocked to the swamps and estuaries of Florida, where they died.

During the risings and sinkings of Florida, the author maintains that leaching by water caused denudation and solution of the highly phosphatic limestone, bicarbonate of lime being carried away in solution and phosphate both in solution and suspension. In the stiller waters of the estuaries, etc., the phosphate of lime in suspension was deposited as an alluvial secondary deposit. At certain places, the water in the streams and rivers, apparently, gave up the phosphate in solution, phosphoric being replaced by carbonic acid. The result is evidently the irregular deposits of rock, boulder, or high grade phosphate of lime.

The great similarity between the appearance of the hæmatites of Virginia and the hard rock phosphates of Florida was first pointed out by Mr. N. H. Darton, and the identity is very striking.

The phosphate carried in suspension, etc., was deposited in the back waters of the rivers and in the shallow streams, forming the soft rock or phosphate. With this were the bones of the beasts mentioned above. "Then, in some not understood way, segregation and accretion took place, the richer phosphatic nodules attracting to themselves the phosphate within the radius of their attraction, and thus was formed the 'nodular' phosphate rock, which consists of rich concretionary nodules in a poorer calcareous matrix mixed with bones and teeth. There is no dispute as to the immediate origin of the pebble phosphate, river and land. The nodules and the bones, mostly rounded by attrition, were washed down by the rivers and distributed in their ever-changing beds with sand and mud. The nodules and the bones form the gravel in the banks in the river beds, the 'river pebble' being merely the present result of river action." J. W.

* *Engineering and Mining Journal* (New York), August 23, 1890.

NAPHTHA IN AUSTRIAN GALICIA.

Die Naphtafelderin Wietrzno. *By* CLAUDIUS ANGERMANN. *Jahrbuch der k. k. Geologischen Reichsanstalt*, 1889, *vol. xxxix., pages* 281-288, *with six sections and one map in the text.*

The naphtha occurs in crevices in Eocene sandstones, which lie in an anticlinal beneath the bituminous "menilite shales." The sandstones were fractured when the folding of the strata took place, whilst the more flexible shales were bent but not broken.

The Wietrzno naphtha works are situated at a height of about 980 feet above sea-level, and the gas wells go down to 822 feet, so that the actual height of the natural gas reservoirs above sea-level is about 164 feet. The pressure of the gas at bank is 2 to 3 atmospheres, while at the bottom of the shaft it attains 15 atmospheres.

The author shows that the lie of the beds is such that the gas-bearing deposits cannot extend very far to east, north, or south of the present workings; to the west are the older well-known works of Bobrka. Some shafts were put down by unscientific searchers at points where the beds are too steeply inclined to permit of the gas-bearing sandstones being reached, and in this way a sum of £17,000 was lost. ——— O. S. E.

PETROLEUM IN FRANCE.

Découverte de Terrains Pétrolifères dans la Limagne d'Auvergne. *By* P. DUBREUIL *and* J. DE CLERCY. *Génie Civil*, 1892, *vol. xxii., page* 102.

Mr. J. de Clercy concluded that mineral oil would probably be found at a considerable depth in the Limagne d'Auvergne, from the similarity of the geological formations to those on the eastern side of the Vosges, and the Tertiary formations of Galicia. He began to bore on the Fauvel system in September, 1891, near Pont du Château station. This hole went down 720 feet and cut some fissures containing bitumen without reaching the oil-bearing strata.

A second boring was made on the opposite side of the plain at Pont Battu near Riom. This hole reached a depth of 906 feet, and proved the existence of beds of shale impregnated with carburetted gases, light petroleum oils and various oily minerals, a certain sign of the vicinity of free petroleum. This hole passed first through limestone impregnated with viscous bitumen. At 294 feet a bituminous crevice was cut which yielded abundance of bitumen of a lighter and more oily nature. At 333 feet the strata at the bottom pushed up, and filled the hole for 30 feet, the flow of bitumen increased, and the first liberation of inflammable gas was observed. From that time progress was impeded by continual upheavals accompanied by gas. At 636 feet the hole was filled with calciferous sand for more than 300 feet, gas could be constantly lighted at the top, and the bitumen became very light and fluid. Other upheavals of 180 and 240 feet occurred, and the hole was then tubed to a depth of 573 feet. At 645 feet a new outburst filled the hole for 426 feet, and the supply of light bitumen and gas was abundant; from 654 to 870 feet hard limestone was passed through without bitumen or gas, but the water coming out of the hole had a strong smell of petroleum different from the smell of bitumen. This water contained little oil, and was very salt. A series of very hard argillaceous beds were next found, with thin beds of siliceous sand, and at 900 feet there were fresh outbursts both at the bottom and 639 feet down, which stopped the boring.

The hole was cleaned out again, during which operation much oil and gas was given off with the water. The bottom again burst up and the boring was stopped on November 1st, 1892, until stronger mechanical appliances could be erected.

It is concluded that petroleum exists under considerable pressure, which may produce spouting wells, if deeper holes encounter permeable measures. W. N. A.

GEOLOGY OF THE CAUCASIAN (BAKU) NAPHTHA REGION.

Preliminära meddelanden från de kaukasiska naftafälten. *By* HJ. SJÖGREN. *Geologiska Föreningens, Förhandlingar, Stockholm*, 1891, *vol. xiii., pages* 89-110, 223-255, *with sketches in the text and a coloured map.*

The author classifies the strata as follows :—

Local Name.	General Name.	Geological Age.
6. ... . . . 5.	Caspian Aralo-Caspian	Alluvial } Quaternary Diluvial }
4. Baku series 3. Apscheron series	Pontino-Caspian	Pliocene
2. Balachany series	... .	Miocene or Oligocene } Tertiary
1. Sumgait series	... ...	Eocene

1. The Sumgait series includes shales, sandstones, and reddish-brown ferruginous marls. The occurrence of iron ochre, nodules of pyrites, abundant rock-salt, and gypsum, the last named often in enormous masses, is noticed. In the Mahannä district, in conjunction with the anticlinal folding of the rocks, there are mud volcanoes with outflows of naphtha and natural asphalt. Limestone also appears in this district and is, moreover, the characteristic rock of the Beschbarmak massif.

2. The Balachany series is the naphtha-bearing series *par excellence*. It comprises grey to greyish-brown slaty marls, calcareous sandstones, and loose sand-beds, saturated with naphtha or water. Fragments of lignite occur, and carbonaceous plant-remains are found abundantly in a hard grey horizontally-bedded sandstone. In the loose sands, the presence of rounded masses of sandstone, about the size of a man's head or larger, is often noticed. It is presumed that these represent really the *débris* of the calcareous sandstones, whose matrix has been dissolved away by the carbonic acid evolved from the naphtha. These masses are found to be a great obstacle in boring for naphtha. Nodules of pyrites occur throughout the series.

3. The Apscheron series, which in some places is conformable with, in others lies unconformably upon, the Balachany beds, is partly argillaceous and partly calcareous in character. It contains limestones which are exclusively built up of molluscan shells.

4. Upon the Apscheron beds rests unconformably the Baku series, made up of shelly limestones, alternating with marls and sands. Above these come the limestones, sands, and gravels of the Aralo-Caspian series. O. S. E.

THE PETROLEUM INDUSTRY OF BAKU.

État actuel de l'Industrie du Naphte dans la Presqu'ile d'Apschéron. *By* A. LEPROUX. *Annales des Mines*, 1892, *series* 9, *vol. ii., pages* 117-162, *and* 7 *figures.*

The subject is considered under four heads—(1) History of the development of Baku to the present time. (2) Conditions and methods of working. (3) Treatment of the petroleum, employment of the residuals (*mazout*) for the manufacture of lubricating oils, and for heating steam-boilers. (4) Cost of production, transport, taxes, etc. Commercial position of Baku.

1. *Historical.*—About the year 1800 the production of petroleum from Baku varied from 1,500 to 2,000 tons. Baku was definitively annexed to Russia in 1801,

and a monopoly established by the Government. Under this *régime*, which lasted until 1872, the production of petroleum increased slowly but continuously, attaining 25,000 tons in 1872. The suppression of the monopoly and gradual abolition of the taxes caused the production to increase to 333,000 tons in 1878.

Inconveniences arising from over-production and want of means of transport began to be felt about that time. In 1884, the railway from Baku to Tiflis was opened, and the production increased rapidly, exceeding 2,000,000 tons in 1887, and 3,000,000 tons in 1888, in spite of the re-establishment of high taxes.

At the present time, the production is not increasing, but not from exhaustion of the supply. In twenty years the production has become equal to that of Pennsylvania, but the value of the petroleum from the two countries is not the same. The Pennsylvanian petroleum furnishes 70 to 80 per cent. of burning oil, that of Baku only 30 per cent.

The oil-lands may be classified as to ownership as follows :—

1. Lands bought from the State, or granted to individuals, who either retain them, or have let or sold them.
2. Lands belonging to certain villages, communities, etc., rented by agreement for development.
3. Lands belonging to the State, of which a very small portion is leased.

Only about one-half of the oil-lands is available for working, but the production is so active that their exhaustion must only be a question of a few years. There will then remain for development the lands retained by the State, without counting the numerous deposits of the rest of the Caucasus, Turkestan, and Bokhara, which, however, produce less remunerative petroleum.

2. *Working.*—The geological formation consists of two series of Tertiary beds, the lower of which yield the petroleum. The upper series is composed of hard limestones and shales and reaches a thickness exceeding 2,000 feet in some places. The lower, or petroliferous, series is composed of sandstone, argillaceous sandstone, sand and shale, the total thickness of which is unknown, but has been proved to be 2,300 feet near Balakhany and Binagadine.

The difficulties of extraction are caused more by the looseness of the strata than their hardness. The boreholes are continually obstructed by running sand, of which the spouting wells throw up large quantities, and the employment of torpedoes to renew the supply is impracticable,

The choice of position for a well is not guided by any precise or logical considerations. It is not known why some pits yield much more abundantly than others. Of two pits only some yards apart, one may yield thousands of tons per day, and the other hardly enough to pay costs.

The spouting of the oil is supposed to be entirely due to the pressure of gas dissolved in the petroleum, and is supposed to occur when a sort of bell-shaped cavity is formed above the base of the pit tubing, an occurrence altogether local and purely accidental, and impossible to foresee before a site of a well is chosen.

It is only known that the abundance of oil increases with the depth, that it is greatest where the bed is pinched or compressed, that it is generally small where the beds are fissured, and that it is very irregular.

The wells are bored, generally with rods; the American system of boring with ropes has been tried, but was not successful. The diameter of the holes varies from 17 to 24 inches, generally beginning with the latter diameter which is gradually reduced as the hole descends; the holes are tubed throughout. The boring is carried as deep as possible, but rarely beyond 1,000 feet. The cost of a complete installation with a well 800 feet deep is about £3,160, including the redemption of capital. The rate of boring is about 6 feet per 24 hours, with an engine of 10 horse-power.

When a spouting well is produced, a violent outburst of gas takes place, soon followed by petroleum mixed with sand. The pressure of the gas sometimes reaches 20 atmospheres, and there have been cases where not only were the boring-tools blown out of the holes, but even the column of tubes has been raised, and bent by the gases, in some cases leading to the complete loss of the boring. On account of these considerable pressures, the contrivances used in Pennsylvania for collecting the oil are rarely used, and much of the petroleum from spouting wells is lost, if large reservoirs have not been previously prepared. A spouting well, instead of being a source of riches, is sometimes the cause of ruin because of the enormous havoc produced by such a violent inundation. When the boring is stopped before spouting occurs the oil is raised in cylindrical vessels with a valve at the bottom by means of a small steel-rope. It is necessary to clean the hole occasionally from obstruction by sand.

The cost of this extraction (labour and engine-power) is from 12s. 6d. to 16s. a day. Extraction continues so long as the output is sufficient to pay costs, which depends on the productiveness of the hole, and the price of crude petroleum. At present an output of 6 or 7 tons per 24 hours justifies continuance of work.

The richness of the holes varies from absolute sterility to the spouting out of millions of gallons per day. The cost of production therefore varies infinitely. The average production of all the wells at Balakhany may be said to have averaged for some years 50 tons per 24 hours.

3. *Treatment of Petroleum and use of Residuals.*—All the crude petroleum is treated at Baku, in the vast suburb called Ville-Noire, where there are more than 150 factories, some of which (Messrs. Nobel's in particular) are very large.

The oil is conveyed from the wells either in pipe-lines or boats for distances varying from 5 to 7½ miles. A pipe-line installation comprises a pumping-station and reservoirs at each end of the line. Worthington or analogous pumps are employed to pump the oil through the pipe-lines. The reservoirs are of thin rivetted sheet-iron, costing about from 10½d. to 12¼d. per cwt. of oil contained. The pipes are of wrought-iron (cast-iron offering much greater resistance to the oil-flow especially in winter). An installation to pump 800 tons in 24 hours 6¼ miles, with a difference of level of 197 feet in favour of the flow cost £12,666.

The selling price of crude petroleum varies enormously, or from 2s. 4½d. to 12s. 8d. per ton.

The treatment comprises the manufacture of lighting-oil and lubricating-oils, by fractional distillation preceded and followed by purifying operations. Lubricating-oils are made from the residuals after the lighting-oil is extracted, but the major part of such residuals is used for heating purposes.

The processes of manufacture, testing, barreling, etc., are briefly described.

The bulk of the residuals (*mazout*) from the manufacture of lighting-oils as well as that from reservoirs where the crude petroleum has stood so long that the oil has evaporated is used for heating purposes. It is used in all the steamboats on the Caspian Sea, the Volga and its affluents, on all the railways of the south-east of Russia, and in all the factories where it can be obtained, and its use for these purposes constantly increases. It produces no cinders, is easy to apply, and for equal weight more advantageous than coal. The theoretical evaporative power of *mazout* is 16·2 (1 pound evaporates 16·2 pounds), while that of anthracite is 12·2, and consequently an equal weight of *mazout* would theoretically evaporate 33 per cent. more water than anthracite, but while coal only yields 60 per cent. of its theoretical calorific power *mazout* may yield 80 per cent., making an advantage in its favour of 75 per cent. Consequently, whenever *mazout* can be had at a price below 7/4ths that of coal, it is advantageous to use it. It is so easily applied that one man can attend to 6, 8, or 10 boilers fired with it.

The results of trials in stationary boilers in Moscow are given, and a *résumé* of three articles published in 1884, 1889, and 1890 in the *Proceedings* of the Institute of Mechanical Engineers on the use of *mazout* in locomotive engines.

4. *Cost of Production, Transport, Imposts, etc. Commercial Situation of Baku.*—As has been stated, it is not possible to fix the cost of production of crude petroleum, but its selling price varies from 2s. 7d. to 12s. 11d. per ton.

The cost of production of kerosene from crude petroleum at the former price is approximately 9s. per ton. The cost of production of *mazout* is very variable; in the summer of 1891 it was 6s. 4d. per ton.

The lighting and lubricating-oils only are exported in quantity, and nearly all the *mazout* is consumed in Russia. The oils are carried in tank-ships holding 700 to 800 tons, or in cistern-waggons holding 10 or 12½ tons each. Kerosene costing 9s. per ton at Baku is worth 39s. 6d. per ton at Batoum, and *mazout* costing 6s. 4d. per ton at Baku is worth 32s. per ton at Batoum.

The commercial situation at Baku in 1891 was very unfavourable, and more than 70 factories were closed during that year. In Russia, Russian petroleum can hold its own, owing to protective duties, but abroad it cannot compete favourably with American oils, and the industrial conditions at Baku are adverse to its success in the struggle.

The methods of working are imperfect, the petroleum s badly collected, much being lost by drainage and evaporation. The processes of distillation and refining compare unfavourably with American methods, and Russian petroleum only yields 30 to 40 per cent. of kerosene, of which some American petroleum yields 80 per cent.

The geographical situation of Baku is also unfavourable for exportation to Europe: the increasing Russian demands, however, may suffice for its future profitable working. W. N. A.

NAPHTHA IN THE CAUCASUS.

Ueber Naphtha im Kaukasus. ANON. *Berg-und Huettenmænnische Zeitung*, 1892, *vol. li., pages* 287-289.

The probability of the early exhaustion of the naphtha-wells of the Caucasus, predicted by many Russian engineers, is denied by the well-known chemist, Prof. Mendelejew. Comparatively few wells are at work in the present oil-district, and these of shallow depth, and, in addition, new oil-finds continue to be made farther away. While the American wells are about 25,000 in number, and reach depths of 1,600 feet, not more than a few hundreds are at work in the Caucasus, and their depths do not exceed 700 feet. In general, the boreholes yield about 16 tons per day each, and those that yield less than 4 or 5 tons are abandoned.

The whole naphtha district has an area of about 6½ square miles. It is bounded on the south and east by the chalk beds of the Aral-Caspian valley, down under which the naphtha-bearing strata extend. Chalk beds have originally formed a sort of roof over the strata of sandy shale containing the naphtha, and the erosion of a part of this roof by the sea has rendered the underlying strata accessible. To the westward, the district extends as far as the mud-volcano of Bog-boga. To the north, the shale beds crop out in a line extending through the valley of Balachani and the southern shore of Lake Sabratski.

The naphtha-bearing strata consist of coarse and fine sands alternating with slaty, variegated shale, and lie in a gentle slope from south-east to north-west.

About 4,000,000 tons of naphtha a year are produced in the Caucasus, 500,000 tons of this being sent abroad, and a considerable quantity being used on the spot. More than 1,500,000 tons of waste oil remain in the Caucasus, and this, which in other countries would form a considerable source of profit, is in Russia simply thrown away. It is given away gratis for use as fuel on the steamers of the river Volga and Caspian Sea, and in various engineering works in the neighbourhood.

The petroleum distilled from the naphtha is sent inland by the Volga river and by the Trans-Caspian railway.

In Baku, naphtha costs 0·015d., 0·020d., or at the most 0·025d. per lb., and its price in various parts of Russia depends chiefly on the cost of its carriage from there.

In Moscow, it costs 0·25d., 0·28d., and as much as 0·30d. per lb., against 0·30d. per lb. of coal. The Russian tariffs for the carriage of coal are very high, and naphtha will continue to supplant it as a fuel unless the Government takes the matter up.

The amount of naphtha now raised in the world is about $\frac{1}{45}$th of the whole output of coal, and although new sources of supply have been discovered in the Trans-Caspian and other districts, it is not likely that the yield will become considerable enough for it to take the place of the older fuel. **A. R. L.**

PETROLEUM IN INDIA.

(1) *Preliminary Report on the Oil Locality near Moghal Kot, in the Sheráni Country, Suleiman Hills.* *By* R. D. Oldham. *Records of the Geological Survey of India*, 1891, *vol. xxiv., pages* 83-84.

The oil issues from a hard, unfossiliferous sandstone, 600 feet thick, and probably of Cretaceous age. The oil is most abundant near the base, where the rock is porous. The springs are found in the river-bed, and they appear to have been flowing for many years. The oil is of a pale yellow colour, clear, and free from water. Analyses prove that the oil is of high quality. The actual flow of oil at present is 10 gallons per day.

(2) *Report on the Oil Springs at Moghal Kot in the Shirani Hills.* *By* Tom D. La Touche. *Records of the Geological Survey of India*, 1892, *vol. xxv., pages* 171-175, *and two plates.*

These oil springs occur near the village of Moghal Kot in the Sherani Hills. They have previously been reported on, and samples of the oil have been analysed, but the present exploration was made to ascertain whether the oil could be procured in sufficient quantities to render it commercially valuable. The oil springs are found in a deep, narrow gorge, cut by the river Toi through a ridge of hard, fine-grained quartzose sandstone, overlain by massive limestone. The oil issues close to the water's edge, and most copiously near the base of the quartzose sandstone. The points of outflow seem to be determined by the existence of beds of shale intercalated in the sandstone. On issuing from the rock the oil is limpid, slightly yellow, and opalescent. From a rough calculation, it appears that one hole would yield a gallon of oil in 4½ hours, while another would take 14 hours.

Although the ridge of rock yielding the oil extends for 30 miles to the north of the river, the occurrence of the oil itself seems restricted to this spot. The peculiar structure of the rocks in the gorge above Moghal Kot facilitates the escape of oil where the springs occur.

The practical conclusion arrived at by the author appears to be that the present flow of oil might, by suitable borings, be sufficiently increased to make it commercially valuable.

(3) *On Mineral Oil from the Suleiman Hills. By* THOMAS H. HOLLAND. *Records of the Geological Survey of India*, 1891, *vol. xxiv., pages* 84-97.

This paper records an analysis of Mr. Oldham's samples (see previous paper), which are inferior to those previously analysed. The paper also contains a valuable bibliography of the petroleum resources of the Punjab and Baluchistan areas.

(4) *Second Note on Mineral Oil from the Suleiman Hills. By* THOMAS H. HOLLAND. *Records of the Geological Survey of India*, 1892, *vol. xxv., pages* 175-180.

Analyses of two samples of the oil from Moghal Kot collected by the previous writer are as follows:—

Specimen A was a deep yellow, mobile liquid, slightly turbid. The specific gravity, at 60 degs. Fahr., was 0·819. The flashing-point (Abel test) was 75 degs. Fahr. The crude oil contains 87½ per cent. of illuminating oil.

Specimen B is a clear, rich straw-coloured liquid. The specific gravity, at 60 degs. Fahr., is 0·811. The flashing-point is 64 degs. Fahr. The crude oil contains 84 per cent. of illuminating oil. There would be a slightly greater waste in preparing oil B for the market. Both samples are of very high value, so that, if occurring in sufficient quantity, no foreign oil could compete with it. G. W. B.

PETROLEUM IN PERSIA.

Note sur les Gîtes de Naphte de Kend-é-Chirin. By J. DE MORGAN. *Annales des Mines*, 1892, *series* 9, *vol. i., pages* 227-238, *with sections in the text.*

The author sketches briefly the physiography of the hilly country between Hamadan and Zohab, a district which forms part of the system of parallel ranges whose height drops gradually from the summit-level of 7,000 feet at the Zagha Pass to 150 feet at Bagdad, in the plains of Mesopotamia. Hamadan lies within a zone of igneous rocks; west of this, fossiliferous Jurassic strata were observed, overlain (apparently in regular succession) by Cretaceous rocks, Nummulitic limestones, and a great series of later Tertiaries with naphtha-bearing beds. The naphtha zone forms a long band, running north-west and south-east from Kerkuk (in Turkish territory) to Kasrashirin, near Shahku. The existence of prolongations of the band north and south of these localities is not improbable.

However that may be, the author's attention was specially devoted to the deposits of Kandashirin, situated at a height of about 1,560 feet above sea-level in the central portion of the naphtha zone, and distant about 95 miles from Bagdad. The immediately surrounding country is rather flat, but there are a few low hills of soft Tertiary rocks. The naphtha occurs here in an anticlinal of unfossiliferous post-Eocene sands and marls, so far as it has at present been tapped; but the author shows good reason for the hypothesis that the main reservoir really lies in one of the synclinals north and south of the anticlinal fold.

At Kandashirin, a little river cuts through the crest of one of the anticlinal folds, so exposing the petroliferous marls beneath. These beds incline to the right and the left of the crest, which runs west 12 degs. north, and their dip varies from 75 degs. at the line of fracture to 18 degs. at a distance of 5,000 feet (1,500 metres); farther on the beds become horizontal, and afterwards rise again. Excepting where exposed by the stream, the marly beds containing the petroleum are entirely covered by superior strata, certainly more than 8,000 feet (2,500 metres) in thickness, and devoid of fossils. The uppermost marly bed contains strings of ozokerite,

varying in width from 0·10 inch (2 or 3 millimetres) to ½ or ¾ inch (15 or 20 millimetres), which have been produced by the infiltration of the mineral oils into the fissures of the marl; these the author considers to be of no practical value.

Numerous remains of shallow wells are visible, and the Kurds have cut a deep trench and sunk two wells, each of about 30 feet (9 metres) in depth. These fill up with salt-water and naphtha, and are emptied every four or five days by means of a cord and bucket, each operation yielding about 50 gallons (250 litres) of crude oil and a large amount of brine—the latter being evaporated down to impure salt, with a strong odour of petroleum, which sells in the country at about ½d. per pound.

The crude oil is taken on mule-back some 10 miles to the village of Kasra-shirin. It is very fluid, green, has a strong odour, and sells crude at about 1d. per pint (1¼d. per litre). It is then refined, when it fetches about double.

The salt produced from the brine in the evaporating-cabins is very bad and strongly tainted with petroleum. In short, the methods of working are of the most primitive description, and only 10 to 12 men are employed. The Kurdish wells have only touched the surface of the deposit; they do not, therefore, go down to the impermeable strata which form the bed of the subterranean naphtha-lake.

The author, in conclusion, points out that the petroleum from Baku and from far Pennsylvania, which now monopolizes the markets of China, India, Persia, and Turkey, may at some future date be displaced by the petroleum from the above-described deposits, the shipping port for which would be Bagdad.

G. E. C. & O. S. E.

SAFETY-CATCH FOR PUMP SPEARS.

Fangvorrichtung für Pumpengestänge. *By* J. Sprenger. *Berg- und Huettenmænnische Zeitung*, 1888, *vol. xlvii., pages* 205-207, 223-226, *and* 231-235, *and two plates.*

In order to reduce the damage done by falling pump-spears in case of their breaking, it has been the practice in some pits to fit catches at intervals in their length in the shaft. The fall then depends on the vertical distance apart of the catches, these having to be the stronger and heavier the wider the intervals. In the König mine, near Neunkirchen, a method is employed by which the broken spears are caught before they have gathered momentum, and the engine brought to a stand at once, and for this system the fittings are lighter and less clumsy than those usually adopted.

There is an upper length of 220 feet of wooden spears, and a lower length of 260 feet of iron ones; and there are clutches fitted at about 60 feet from the upper end and 50 feet from the lower end. To each side of the spear at each of these places is a spindle with eccentric wheels at its ends, in the perimeters of which are deep grooves corresponding in shape to the edges of an iron standard fixed between them opposite the centre of the spear. Between the two systems of wheels is a strong rope; from the upper system to the top, another of about half the strength; and from the lower system to a cleat near the bottom of the lower spear is a third rope of about a quarter of the strength of that first mentioned.

The attachment of the ropes is similar in the two systems. Each spindle has a short crank which is turned inwards and attached to the obtuse side-angles of a loose parallelogram made of four iron bars, the ropes being attached above and below to its acute angles.

Close to the top of the shaft the rope is reeved through two eyes on a plate fixed to the spear. A nut between the two keeps the rope taut below, and in case

of a breakdown will fly up against the upper one, there being a few inches of play between. The ropes are made taut by tightening screws, and so arranged that the cranks on the spindles point inwards and keep the eccentric wheels with their smaller diameters towards the fixed standards between them. The upper rope is carried first over the end of the pumping beam and then inwards over a pulley, and at its end hangs a heavy weight which, in case of accident, will fall on a lever and shut off the steam. In case a spear breaks the weak lower rope will break also, and this setting free the parallelogram, the lower eccentric wheels are turned by their own weight till they grip the fixed standard and thus arrest the falling spear. Meanwhile the upstroke continuing, the next weakest upper rope will give way and allow the weight to fall and shut off the steam. The strong middle rope does not break, but being kept in considerable tension before the accident will, by returning to its original length, help to bring both upper and lower eccentric wheels into position to grip the standards. The standards, about 10 feet long, are fixed at their upper and lower ends on to beams lying across the shaft.

For deeper pits more than two clutches may be required, and these can be arranged in a similar manner. In the König mine the arrangement was practically tested, by cutting first the lower rope and then the upper one, while the pump was at work. The spears were at once arrested, and the engine brought to a standstill.

A. R. L.

BURMAH RUBY MINES.

Note on the reported Namsèka Ruby-mine in the Mainglôn State. *By* FRITZ NOETLING. *Records of the Geological Survey of India*, 1891, *vol. xxiv., pages* 119-125.

The original matrix of the ruby is a crystalline limestone, but the mine is worked in river alluvium. Valuable rubies are said to have been found at this mine, but a trial of three days of eight hours each, with twelve coolies, produced no result. Since any rubies which may have been found in this mine have probably been washed down from the ruby-mine district by the river Mogaung, other deposits might be expected along the course of this stream. G. W. B.

EXPERIMENTS WITH SAFETY-LAMPS.

Expériences sur les Lampes de Sûreté. Rapport présenté à la Commission du Grisou au nom de la Sous-commission chargée des recherches expérimentales. Annales des Mines, 1892, *series* 9, *vol. i., pages* 47-66, *and plates III. and IV.*

The experiments were made by Messrs. Mallard, Le Chatelier, and Chesneau at the École des Mines, in a laboratory fitted up for the purpose. No attempt is made to compare the inconveniences or advantages of the lamps tried except as to safety and lighting power.

I. DESCRIPTION OF THE APPARATUS.

One apparatus was used for examining the behaviour of safety-lamps when placed in a stationary or slowly-moving explosive mixture of air and marsh gas. The formene was stored in a gasometer of 212 cubic feet capacity, from thence it passed to a meter. A fan worked by a gas engine at the same time passed air through another meter. The mixed gas and air passed by a pipe into the base of a square box in which the lamp was placed. This box was glazed on four sides for observation, and

fitted with a small chimney at the top. Regulating cocks allowed the flow of gas and air through the meters to be adjusted. By another pipe and cock, lighting gas could be substituted for formene. In this apparatus a safety-lamp could be kept for several hours in a known mixture of air and formene or air and lighting gas.

The second apparatus was used for testing lamps in mixtures of air and lighting gas moving at great velocities. The lighting gas passed through an 800-burner meter, into the base of an upright pipe into which air was also forced by a fan. The mixed gas and air ascended the vertical pipe and passed into a short horizontal box in which the lamps were suspended opposite a window for observation. This box was movable on a horizontal pivot fixed at its junction with the vertical pipe, so that it could be moved or jerked by a cord passing over a pulley to the observer. The velocity of the explosive mixture was measured by an anemometer. The proportion of gas was measured by the meter. The composition of the mixture was usually the most explosive possible. The velocity could be raised to 20 or 23 feet per second, corresponding to 33 or 40 feet in the area occupied and restricted by the lamp.

II. Result of the Experiments.

Modified Boty Lamp.—A modification of the Boty lamp, having the gauze cylinder replaced by a cylinder composed of flat iron rings placed one above another and maintained 0·039 inch apart. Its weight, without oil, was 2·96 lbs. The inventor hoped that the large surface of the flat rings between which the air enters and leaves the lamp would so cool enflamed gas as to prevent the passage of flame even in currents of high velocity. When placed in a mixture of maximum explosiveness the wick-flame was extinguished, but the gas continued to burn in the interior of the ringed cylinder which became red-hot. With a velocity of 8·03 feet per second, (12·13 feet at the lamp) explosion did not occur at the end of 10 minutes; with higher velocities the results were different, as follows:—With a velocity of 10·66 feet per second (15·41 feet at lamp) explosion in 30 seconds; with a velocity of 19·18 feet per second (27·88 feet at lamp) explosion in 5 seconds: with a velocity of 25·58 feet per second (34·27 feet at lamp) explosion instantly. The hope of the inventor was not realized, as it only required a slight increase of velocity to pass the flame.

Pieler Lamp.—This lamp has a double gauze and a shield with a longitudinal window for observation, closed by a movable shutter, and fed with alcohol. The weight when empty was 3·78 lbs. With a velocity of 8·03 feet per second (15·35 feet at lamp) the lamp with shield and 2 gauzes, the window open and directed towards the current, the interior gauze reddened strongly; but explosion was not produced at the end of 5 minutes (the lamp was violently agitated in all directions by sharp rotary movements of the box by the cord). With the same velocity, but the lamp with shield and 1 gauze only, the same result was obtained. Without the shield, but with 2 gauzes, explosion resulted in 15 seconds. With a higher velocity of 23·61 feet per second (45·10 feet at lamp), the lamp with shield and 2 gauzes, window turned towards current, lamp agitated in all directions, the gauzes reddened strongly, but no explosion after 5 minutes. Lamp with shield and 1 gauze same experiment repeated; no explosion occurred at the end of 5 minutes, but one happened at the moment when the gas was shut off.* flames produced by the combustion of alcohol vapour escaped with violence by the lower range of holes in the shield. These experiments show that the Pieler lamp, which is still regarded with

* Explosions produced at the very moment when the gas was shut off were observed rather frequently during the experiments. It may perhaps be explained thus:—The proportion of air being sharply increased the red-hot wires of the gauze enter into combustion and ignite the still inflammable gaseous mixture.

much suspicion, is safer than is generally supposed, and that the shield in this lamp is a great element of safety, even when it is pierced by a long but narrow window.

Cambesèdes Lamp.—This lamp is specially designed to give a good light; it is constructed on the principle of an argand lamp, with an oil reservoir surrounding about $\frac{1}{3}$ of the glass, and its lighting power is about 1·25 candle-power.

The oil is kept at a constant level, and the lamp was devised to give more light than the ordinary type of lamp; that aim appears to be successfully fulfilled, as proved by the photometric experiments. The weight, when empty, is 3·33 lbs., and, with oil, 3·63 lbs. When placed in a mixture of air and marsh gas the wick-flame is extinguished and the gas does not burn in the interior. In a mixture of air and lighting-gas of maximum explosiveness moving at 9·67 feet per second (18·20 feet at lamp), in one experiment the lamp went out in a few seconds without lighting the gas inside; in two other experiments, after the extinction of the wick-flame the gas continued to burn inside, resulting in a heating of the lamp, which caused an abundant escape of oil from the reservoir; the experiments were stopped after 2 minutes. With a velocity of 18·36 feet per second (35·09 feet at lamp), exactly the same results wese obtained.

The tests in presence of explosive gaseous mixtufes showed that in a rapid current the flame is not passed outwards, but sometimes persists within the lamp, and thus causes oil to escape from the reservoir.

*Thorneburry Lamp.**—This is a petroleum lamp, the weight, empty, being 3·92 lbs. The petroleum used is relatively heavy, its specific gravity being 0·831 at 60 degs. Fahr., flashing (Abel test) at 256 degs. Fahr. The lamp is of excellent lighting-power, the maximum light after burning 15 minutes being 1·44 candle-power, and, after 40 minutes, 1·20 candle-power. The regulation of flame is delicate, depending on temperature. When the lamp is placed in a mixture of air and fire-damp, no cap is visible, on account of the light. The lengthening of the wick-flame gives very good indications, with 1·5 to 3·5 per cent. of fire-damp. With a greater proportion, the lamp smokes and the flame lengthens considerably. For these indications the flame must be first regulated in fresh air. In a current of air and lighting-gas at a velocity of 18·20 feet per second (36·40 feet at lamp), the lamp being in its normal state, the wick-flame extinguished at once, and the gas was not ignited inside. With the lamp without interior gauze, the wick-flame was extinguished, the gas burnt inside for 3 or 4 seconds, and then went out (two experiments). With the lamp without gauze, and the proportion of gas gradually increased to maximum, the flame lengthened progressively, smoked, but was not extinguished till 6 per cent. of gas was attained; with more than 6 per cent., the wick-flame was extinguished, and the gas burned inside the lamp for 2 or 3 seconds only; and the interior glass was broken, but not separated, when the lamp was kept 1 minute in the mixture of 6 per cent. This glass being thin, it might break and fall in pieces. The lamp was therefore tried without the interior glass, with a velocity of 21·97 feet per second (44·28 feet at lamp), the wick-flame was extinguished at once; the explosive mixture burnt in the gauze, which heated considerably; but there was no explosion at the end of 5 minutes. The lamp tried without gauze or interior glass gave immediate explosion.

The Thorneburry lamp possesses a high degree of security, and gives a very superior light. Till now, petroleum has not been considered a safe oil for safety-lamps. There is reason to consider whether there is serious danger when a heavy and not very volatile oil is used. Only an explosion in the oil-vessel is to be feared, which appears very unlikely with heavy oil.

* *Trans. Fed. Inst.*, vol. iii, page 226.

Fumat Lamp.—The latest modification of this lamp was used, in which the air enters through a series of holes in one side only of the lower part of a shield above the glass-cylinder, and the smoke escapes through the upper holes. The air passes into a sort of box, which hides half the glass, and after traversing the gauzes, reaches the flame. The lamp burns well in horizontal or descending currents, but not in ascending currents in the absence of gas. It gives a good light, equal to 0·87 candle-power. In a mixture of air and marsh-gas of maximum explosiveness, it is at once extinguished. With 6·5 per cent. of marsh-gas, it was only extinguished after 2 minutes. In a mixture of air and lighting-gas, at a velocity of 10·99 feet per second (37·39 feet at lamp), three experiments gave identical results; the wick-flame was extinguished at once, and the mixture burnt in the gauze from 10 to 20 seconds only, even with the holes in the shield facing the current. The Fumat lamp therefore possesses all the desirable guarantees of safety.

Marsaut Lamp.—After an accident in St. Etienne, in 1889, suspicion was cast on the Marsaut lamp, and it was proved that when placed in a mixture of air and lighting-gas, the gas continued to burn in the lamp after the extinction of the wick-flame, and fear was expressed that if the lamp so remained for long, the gauze might become so hot as to allow passage of flame, but Mr. Marsaut made experiments proving that the fear was groundless. In order to study this phenomenon, a large sheet of mica was inserted in the shield, so that the gauze could be seen. In a mixture of air and 10 per cent. of marsh-gas, introduced at the rate of 61 cubic inches per second, the wick-flame went out and the gas burnt within the gauze. The lamp was left in this condition several times for periods varying from 1 to 3 hours, and the gauze never sensibly reddened or altered; it was therefore proved that the imagined danger does not exist. In a current of air and lighting-gas of maximum explosiveness, at a velocity of 23·61 feet per second (38·24 feet at lamp), with the lamp in its normal state (1 shield and 2 gauzes), the wick-flame was extinguished; the gas continued to burn within the gauze, which became red, but only for a fraction of an inch at the base, and the lamp being vigorously agitated, no explosion occurred at the end of 5 minutes. With the lamp with shield and 1 gauze the same conditions gave the same results, proving that with only one gauze the Marsaut lamp was perfectly safe.

Modified Marsaut Lamps.—Experiments were made with the Marsaut lamp, modified by having a large opening cut in the shield 3·03 by 1·77 inches, and with another lamp with holes in the top and bottom of the shield opposite the gauze, and with other modifications. These alterations were proved by experiments to interfere materially with the safety of the lamp. W. N. A.

THE CUVELIER LOCK FOR SAFETY-LAMPS.

Considérations sur la Question de la Fermeture des Lampes de Sûreté. By JOSEPH GOFFIN. *Publications de la Société des Ingénieurs sortis de l'École Provinciale d'Industrie et des Mines du Hainaut*, 1892, *series* 3, *vol. i., pages* 208-215, *and one figure.*

In the new form of the Cuvelier lock, as in the old form,* the lamp is fastened by connecting its upper and lower portions by a bolt. This bolt is vertical and is connected with a spiral spring tending to draw it down. At its upper end is a groove, into which fits the extremity of a horizontal lever, so as to keep it raised, the lever being kept in position by a strong spring. Against the lever impinges a socket, into which fits a rod with a hole down through it, which can be connected

* *Trans. N.E. Inst. Min. Eng.*, vol. xxxvi., page 51.

with a hydraulic accumulator. The pressure so exercised forces the socket forward, which reacts on the lever so as to free the head of the bolt; and this is forced down by a spring and the lamp unfastened. It can be refastened by simply raising the bolt, so that the end of the lever catches into the groove around its head. G. E. C.

RELIGHTING SAFETY-LAMPS IN COLLIERIES.

Note sur le Rallumage des Lampes de Sûreté dans les Mines. *By* JOSEPH GOFFIN. *Revue Universelle des Mines, etc.*, 1892, *series* 3, *vol. xviii.*, *pages* 39-49, *and plate II.* (*eight figures*), *and Publications de la Société des Ingénieurs sortis de l'École Provinciale d'Industrie et des Mines du Hainaut*, 1892, *series* 3, *vol. i.*, *pages* 173-187, *and plate XIII.*

One of the great faults of the Mueseler lamp is the ease with which it is extinguished. Exact statistics from several collieries show that 20 per cent. of the lamps in use are accidentally extinguished daily. This extinction causes great loss of time, as the lamps have to be taken to distant places to be re-lighted, and it tempts the workmen to open their lamps surreptitiously.

In order to overcome this inconvenience several methods have been devised for the interior re-lighting of lamps. The electric re-lighter of Messrs. Durant and Herbert (1881) was the first, or one of the first of these inventions. Two enamelled metal rods pass through the oil-vessel of the safety-lamp, one on each side of the wick-tube, and a thin platinum wire passing above the wick unites the two rods. The bases of the rods terminate in two buttons, intended to be applied to the poles of an electric battery so as to heat the platinum wire. To re-light the lamp the wick is raised till it touches the wire, and the bases of the rods applied to the poles of the battery; this renders the wire incandescent, and so relights the lamp. This system was adopted in one colliery, but its use has been discontinued.

The re-lighting lamp of Messrs. Mori and Rhodes is based on the same principle as the last lamp, but the details of construction are improved. In this lamp the platinum wire is of a horse-shoe shape, and is placed so as to touch the wick. The metal conductors are each made in two portions, which are kept apart by a spring; when connected with the electric battery the circuit is completed, the wire rendered incandescent, and the wick re-lighted.

The system of re-lighting lamps by electricity does not completely meet the difficulty, because the lamps require to be taken to the place where the battery is kept, and a number of batteries or accumulators have to be placed in each of the various districts of a mine.

These inconveniences are entirely overcome by the use of internal re-lighters, by which the wick is lighted either by fulminating-caps attached to a band of strong paper or by matches.

The Wolf benzine lamp is re-lighted by fulminating-caps; a narrow roll of paper to which discs of the fulminate are attached is carried in the base of the lamp, and by mechanical contrivances the paper is drawn past the wick, and a disc of fulminate ignited in proximity to it. This system is only applicable to lamps fed with volatile mineral oils such as benzine.*

The writer, however, strongly objects to the use of such volatile oils in safety-lamps.

The Catrice re-lighter† may be used in lamps burning ordinary oil. The lamp is

* *Trans. N.E. Inst. Min. Eng.*, vol. xxxiv., page 291, and plate XLI.

† *Ibid.*, vol. xxxvii., abs., page 54.

re-lighted by means of a match, which, by a mechanical arrangement described, is shoved up a tube alongside the wick, and ignited in its passage by friction. A small barrel, somewhat like the barrel of a revolver, placed in the oil-vessel, contains 13 matches, tipped with an easily-inflammable chemical composition, and these are used one after another. W. N. A. and G. E. C.

TOMMASI ELECTRIC SAFETY-LAMP.

Nouvelle Lampe Electrique de Sûreté. By D. Tommasi. *Revue Universelle des Inventions Nouvelles. Edition A*, 1892, *pages* 238-239, *and figure.*

In the Tommasi electric safety-lamp, the glow-lamp is enclosed in an air-tight glass cylinder, closed below by the lower part of the lamp, and above by a cover fitted with a stop-cock. In the lower part of the lamp is a pair of bellows full of air, so arranged that when full it cuts off the electric current. To light the lamp a quantity of air is forced through the stop-cock in the upper cover. The tension thus produced depresses the bellows and establishes the current. The lamp is extinguished by opening the stop-cock, which relieves the pressure, the bellows then rise, and cut off the electric current. The current is similarly cut off when the glass-cylinder is broken. If the glass globe of the glow-lamp be broken, the air inside the glass-cylinder expands, the bellows open and cut off the current as before. G. W. B.

THE WOLF BENZINE SAFETY-LAMP.*

Ueber einige praktische Erfahrungen beim Gebrauche der Wolf'schen Benzin-Sicherheitslampe. By P., *mining engineer. Berg-und Huettenmœnnische Zeitung*, 1891, *vol. l., pages* 193-195.

The facility with which the closed Wolf lamp (in its present perfected state) can be immediately re-lighted, and thus made use of in rescuing miners in case of serious accidents, has been proved at the explosion which took place in the Dreifaltigkeit pit, Polnisch-Ostrau, on January 3rd, 1891. The workmen were able to find their way to a place of safety, thanks to the light afforded by their lamps (which had been extinguished by the rush of gases, but which they had been able to re-kindle at once).

In addition to furnishing a stronger and steadier light than is the case with ordinary oil safety-lamps, the Wolf lamp gives more timely and more decisive indications of the presence of fire-damp; the flame can be so adjusted that a practised eye may detect the presence of 1 per cent. of the gas in the pit. But these advantages can only be reckoned with on condition of using perfectly pure, double-distilled benzine—a circumstance often overlooked in practice. O. S. E.

SALT-MINING IN THE AUSTRIAN ALPS.

Der Salzbergbau in den österreichischen Alpen. By August Aigner. *Berg-und Hüttenmännisches Jahrbuch*, 1892, *vol. xl., pages* 203-280, *with three plates.*

After a preliminary paragraph, implying that no complete description of salt-mining in that part of Europe has been published since the appearance of Mr. Ritter von Hauenfels' paper in the *Leoben Mining Journal*, in 1853, the author proceeds to consider:—

* *Trans. N E. Inst. Min. Eng.*, vol. xxxiv, page 291, and plate XLI.

I. *The Saline Formations in the Alpine Districts.*—These range from Maria Zell, in the east, to Hall (Tyrol), in the west, the rock-salt occurring mostly in the Werfen Shales, the lowest division of the Bunter Sandstone series of the Trias. The measures worked for salt are called the Haselgebirge, and are, in point of fact, a chaotic jumble of rock-salt, saline and gypseous clays, polyhalite, anhydrite, soluble magnesium and sodium sulphates, hepatite, werfen, and other shales, etc. The rock-salt is extremely variable in colour and in structure; the percentage of pure sodium chloride in it varies from 96 to 100. The polyhalite occurs in fibrous plates of an inch or more in thickness, the undersides of which have generally a covering of clay; an average of chemical analyses shows that it contains 52·9 per cent. of calcium sulphate, 9·6 per cent. of magnesium sulphate, 12·4 per cent. of potassium sulphate, 5·4 per cent. of sodium chloride, ½ per cent. of sodium sulphate, 10 01 per cent. of aluminium silicate, and 4·2 per cent. of water.

In the Kammergut the principal mines are those of Aussee, Hallstatt, and Ischl. That of Aussee is the richest, the yearly brine-production averaging 590,000 hectolitres (13,000,000 gallons). In it 13 adits have been opened, and the greatest depth attained, which has not yet touched the footwall, is 544 metres (1,785 feet); the hanging wall is Zlambach marl, a fossiliferous stratum, containing about 3 per cent. of calcium carbonate and 7·5 per cent. of the sulphate. In the Hallstatt mine the hanging wall appears to be a breccia, in which are commingled the Zlambach marl, Hallstatt, and Dachstein limestones, and other strata broken up by the uprise of an intrusive igneous mass—identified as melaphyre. Here 15 adits have been opened; the greatest depth attained is 450 metres (1,476 feet), and the annual brine-production averages 1,900,000 hectolitres (42,000,000 gallons). In the Ischl mine, where a depth of 723 metres (2,372 feet) has been reached, though the footwall is as yet unknown, the strata occur in undisturbed succession. The adits are 14 in number, and 700,000 hectolitres (15,000,000 gallons) of brine are produced yearly.

Turning now to another district, the Hallein mine is worked in an overfold, the extent and depth of which are unknown. The strata here are reckoned to be about as rich in salt as those of Ischl; the annual brine-production averages 834,000 hectolitres (18,500,000 gallons). It is noticeable that a large proportion of magnesium chloride is present in the salt deposit. The saline measures of Berchtesgaden are practically a continuation of those of Hallein, and the brine produced from them annually amounts to 1,200,000 hectolitres (26,500,000 gallons). In that neighbourhood salt-mining is of very ancient date, and the industry was carried on under the rule of the mediæval bishops of Salzburg and provosts of Berchtesgaden. The Hall (Tyrol) mine is worked in the Keuper—that is, at a far higher geological horizon than the other mines, and the saline deposit is very poor, the maximum percentage of salt being 35. The strata appear to have been much affected here by dynamic influences, one proof of such action being the occurrence of galena in contact with the rock-salt. The greatest depth struck is 300 metres (984 feet), and the annual brine-production averages 523,000 hectolitres (11,500,000 gallons).

The author discusses at considerable length the question of the origin of the salt-deposits, and enumerates his conclusions in the following summary of the phases of formation:—

1. Evaporation (disturbed by earth-movements) of concentrated mother liquors, accompanied by vast accession of heat from the underlying strata, or even by atmospheric evaporation; and horizontal deposition of the clays.

2. Squeezing and folding of the saline beds at a later epoch, under enormous pressure.

3. Dynamic action (eruptive rocks) due to causes partly active at the surface, partly at great depths, and contemporaneous destruction of the entire complex of strata—forming the Haselgebirge measures.

4. Partial folding of the mass of the Haselgebirge, by means of lateral pressure ensuing on mountain-formation, and dynamic influences continuing active down to recent times.

II. *Method of Working the Salt-deposits.*—The salt layer, being presumably first proved by a borehole, a shaft sloping at an angle of 40 degs. is driven down to the required spot from the gallery or upper level; it is provided with a stairway and hand-rail and pipes for taking the fresh water down to the salt. The chamber cut in the rock-salt preliminary to conversion of the rock into brine is usually elliptical. Preference is given to this form, because in measures which are unequally saline the short axis of the ellipse can be made parallel to the strike of the rock-salt *débris*, with a view to a future cutting-off of the chamber. As to the methods of bringing the brine from the pit bottom, the author describes some which have a purely academic interest, for they appear to be discarded nowadays; and even of the others he remarks, on page 246, that they are all antiquated, except the Ebenwehre or horizontal system, which is the cheapest and simplest. (It is illustrated in Fig. 4, of Plate V., in the original paper.) The mechanical details in this section of the paper can hardly be summarized in the absence of the descriptive illustrations, but they do not seem, on the whole, likely to be of much practical value for the English reader. The paper, so far as published, winds up with a series of theoretical considerations, largely mathematical, on changes of volume (in salt and water) consequent on the production of brine. **O. S. E.**

SALT INDUSTRY IN ITALY.

Eine neue Saline in Italien. By F. B. *Berg-und Huettenmœnnische Zeitung*, 1892, *vol. li., pages* 347-348.

The rocky island of Ischia in the Gulf of Naples is of volcanic origin, and is well-known for its hot mineral springs, the town of Casamicciola on its northern slope being a favourite watering-place. The highest hill on the island is the steepsided Mount Epomeo, and between its southern base and the sea there is a gently inclined sandy beach called "dei Maronti" about a mile and a quarter long by 130 feet broad, which is constantly warm even below low water-mark: summer and winter it has a temperature of 122 degs. Fahr. at a foot below the surface, the heat increasing with the depth to 221 degs. Fahr. at 39 inches down. Experiments at different points, in different seasons, and in different years, have given nearly the same results. The waters of two brooks, flowing down from the mountain over this expanse, have temperatures of from 160 to 210 degs. Fahr.

On the strength of the above experiments, a company approached the Government for a concession to win salt from the sea by the aid of tanks embedded in the "dei Maronti" sand. It was shown that sea-water contained in a zinc box 39 inches square imbedded in the hot sand, evaporated within three days without boiling or bubbling, leaving all its contents of salt behind dry, and the experiment was repeated in 100 different places with a like result. The Government agreed to grant the concession, and preparations were being made to carry on the work with large tanks 39 inches deep, having their upper edges level with the sea, when the destruction of Casamicciolo by an earthquake stopped the enterprise, and the concesssion lapsed.

It was calculated that one ton of common salt, which in Italy costs about 42s., could be sold at a profit for 12s. The sea-water round Ischia containing 26½ per cent. of common salt, and 14 per cent. of secondary salt, a basin 39 inches deep with an area of 247 acres would produce 122,000 tons of common, and 64,845 tons of secondary salt. In other Italian salt-works 41 tons of coal are burnt per 100 tons of salt produced, but at Ischia the only work required would be the opening and shutting of the sluices, the collecting of the salt when dried, and the watching of the course of the evaporation.

The temperature being constant and the boiling of the solution impossible, the common salt would first be given off and obtained almost pure. The making of the secondary salt would be easily and cheaply effected by means of the hot streams already mentioned.

The cost of making the salt is reckoned at about 2s. per ton. A. R. L.

THE SALT LAKES OF SOUTH-WESTERN SIBERIA.

Die Salzseen von S-W. Sibirien. By R. Helmhacker. *Berg-und Huettenmænnische Zeitung*, 1892, *vol. li., pages* 233-235.

In the low-lying steppes of South-Western and of the northern part of Mid Siberia is a large tract of slightly depressed country studded with numberless salt-lakes, which are gradually drying up. They are the remains of a large sheet of brackish water which once included the Aral and Caspian seas, these two latter having within a few decades themselves perceptibly decreased in area. The surface formation in these districts is a clayey sand alternating with sandy clay of a light brown or light grey colour, covered with a thin stratum of soil in which flourishes a very profuse vegetation.

The original sea having been brackish, the lakes, as they dry up, become constantly more salt, and deposit in greater or less quantities and proportions common salt (chloride of soda), bitter salt (sulphate of magnesia), Glauber salt (sulphate of soda), and other salts. Many of them deposit a thick crust of salt on their beds, which is collected round their margins in summer when evaporation diminishes their area. Other lakes again are not sufficiently salt-bearing to form such deposits.

The lakes belong, some to the Privy Exchequer of the Czar, some to the Russian State, and others to the Kirghiz.

Common salt is the kind most extensively met with, and this is dredged up or collected round the edges of the lakes and heaped up on land, to be dried and separated from sand and other impurities by the summer sun. It is sent to market in boats on the rivers Irtish and Isym, or in winter by sledges.

The proportion of salt in the water varies very much in different lakes, and a large number of them are not worth working, but these will become more and more salt as they dry up, and may be profitably worked at some future period.

A. R. L.

THE BROKEN HILL MINES, NEW SOUTH WALES.

On the Geological Occurrence of the Broken Hill Ore-deposits. By E. F. Pittman. *Records of the Geological Survey of New South Wales*, 1892, *vol. iii., pages* 45-49.

The geological occurrence of the Broken Hill lode presents many analogous features to the well-known saddle-formation of Bendigo, Victoria. In the latter case the strata consist of Lower Silurian slates and sandstones, contorted into

anticlines and synclines. The saddles are found at the apices of the anticlines, the legs thinning out and disappearing in depth, other saddles being found immediately underneath at greater or less depth. The reefs are associated with narrow intrusive dykes of dolerite, which have doubtless influenced their mineralization.

The Broken Hill district is composed of crystalline gneisses, passing into banded quartzites, micaceous and hornblendic schists, and garnetiferous sandstones, similarly contorted. Their age is uncertain, but, according to the late Mr. Wilkinson, they are as old as Lower Silurian, and probably older. Intrusive dykes of highly-basic diorite extend roughly parallel to the hill on both sides, the hill itself corresponding to one of these anticlines. The ridge consists of the outcrop of the lode—a mass of manganiferous iron ore. Below, in the mine, the lode is found to divide into two branches, dipping east and west respectively. The dividing rock, which has been variously spoken of as horse and intrusion, has not the character of either. Its foliations indicate that it is the cap of an anticline, and suggests that the lode itself is similar to the Bendigo saddles.

This intrusion is found at differing depths in the various mines along the hill, which is another point of similarity with the Bendigo saddles, the rocks enclosing which are contorted into curves along their line of strike as well as transversely.

The Broken Hill deposit, however, differs from the Bendigo reefs in dimensions and character of gangue and mineralization, the latter consisting of auriferous quartz with a little iron pyrites, while the former consists at the outcrop of manganiferous ironstone, succeeded by kaolin and oxidized lead ores, followed in depth by lead and zinc sulphides.

If the author's theory prove correct, both portions of the Broken Hill deposit will probably thin out and disappear in depth—fortunately as yet a remote contingency. On the other hand, it is possible that by boring through the cap of the anticline other deposits might be found more or less vertically underneath.

G. E. C.

DRAINAGE OF SINKING SHAFTS: TOMSON SYSTEM.

Creusement des puits avec épuisement par machine d'extraction, système Eng. Tomson. *By* A. DE V. *Revue Universelle des Mines*, 1893, *series* 3, *vol. xxi.*, *pages* 225-227, *and one plate.*

The No. 1 shaft of the Preussen mine, near Galunen, formerly bearing the name of Gustavus Adolphus, had been commenced in the spring of 1873, but was abandoned in 1875, owing to the sudden irruption of a large quantity of water at the depth of 890 feet, the water flowing out to the surface.

The level of the water having fallen after a certain time, the pit was filled in with concrete to a height of about 150 feet, that is, to a depth of 740 feet from the surface, and it remained in this state until 1891.

There were two winding-engines with two cylinders, 31 inches and 18 inches in diameter respectively, with 61 inches and 31 inches strokes, and a strong steam-winch. The first of these machines was employed in drawing water, the second for sending down the workmen, material, and winding up of the excavated matter; the winch was used for working a movable platform suspended in the shaft.

The main difficulty was to fill the kibble rapidly and completely, and to keep the bottom of the sinking free from a quantity of water not exceeding 50 gallons per minute. This difficulty was overcome by the installation, at some feet from the bottom, of sheet-iron tanks, into which the water-tubs dipped, these tanks being kept full by means of a pulsometer pump suspended at the bottom of the shaft.

The water-tubs were guided by two small steel-wire ropes, which were also used to support the sheet-iron tanks. At the surface these ropes passed over pulleys to the steam-winch.

The water-tanks were connected at the bottom, and were a few inches greater in diameter than the water-tubs, and about 3 feet longer.

The pulsometer pump was placed on the same cradle as that supporting the tanks, or it might be suspended by a special rope from the winch at the surface. Having to raise the water only a small height the pulsometer pump acted excellently under these conditions. The steam was carried to it by a 2¾ inches wrought-iron pipe well covered.

The whole of the installation could be lowered proportionally as the depth of the sinking increased, or raised to the surface again, when it was desired to clear the whole of the pit at any time.

The shaft was opened from a depth of 740 to 850 feet, and the walling completed, drawing at the same time a feeder which finally increased to 300 gallons per minute. R. A. S. R.

THE POETSCH METHOD OF SINKING.*

Ein neuer Erfolg des Pötsch'schen Gefrierverfahrens beim Schachtabteufen. By W. SCHULZ. *Glückauf*, 1892, *vol. xxviii., pages* 1053-1056.

In the middle of October, 1892, the author (then visiting the Lens collieries in Northern France) went down No. 10 shaft, which had been successfully sunk to the coal-measures by the Poetsch freezing-process; and at the same time he saw the process in actual operation in the No. 10 bis shaft, which is just 100 feet distant from the first-mentioned shaft.

The strata through which both shafts were sunk, down to about 138 feet from the surface contain a good deal of water. In descending order we have 32 feet of alternating sands and sandy and marly clays; and then 100 feet of Cretaceous rocks, consisting partly of soft, clayey chalk, and hard, much-jointed, loose chalk with flints, and partly of massive beds of very hard chalk; below this comes an absolutely impermeable layer of clay, overlying a water-bearing blue marl (which yields from 3 to 3¼ cubic feet per minute). The sequence of the strata had been ascertained by means of a boring to a depth of 264 feet.

In sinking shaft No. 10 the ordinary methods were at first used; cast-iron tubbing, 1 inch thick, and 15¾ feet in inside diameter was put in resting on three wooden wedging-cribs, and the space betwixt the iron tubbing and the outer wall of masonry was filled with concrete. The iron tubbing was built up of 39 inches rings, six segments to each ring, and was continued below by a tubbing of oaken timber (about 6 inches thick). The sinking was then carried on by piling, but at a depth of 84 feet from bank, the shaft was flooded and filled with mud through the breakage of a piece of timber on the north side of the shaft. The inrush of water increasing to 875 cubic feet per minute, and the rate of progress of the sinking during the preceding fortnight having been practically *nil*, the engineers decided to resort to the Poetsch process. The pumps were accordingly withdrawn, 8 boreholes were pierced within the area of the shaft, and 20 without it, down into the above-mentioned blue marl, to a depth of 5 feet below the clay which upholds the water and 136 feet from the surface. This was done by a method of cable-boring invented by Mr. Reumaux, the manager of the Lens collieries, and the holes were lined temporarily with tubing which was taken out after the freezing pipes had been

* *Trans. Fed. Inst. Min. Eng.*, vol. ii., page 441, and plate XLI.; and vol. iii., page 1023

placed in position. The freezing liquid was a solution of chloride of calcium cooled by an Osenbrück ammonia-refrigerator; the liquid was led into the freezing-pipes at an initial temperature of 20 degs. Fahr., which was reduced later on to a minimum of 2 degs. Fahr., and was pumped back to the refrigerator at temperatures of 27 degs. Fahr. and 7 degs. Fahr. respectively. The process of freezing went on for 203 days, but the sinking operations were not started till the 228th day. Mr. Reumaux calculated that the beds which were frozen contained about 40 per cent. of water.

The sinking in the frozen mass was carried on partly by means of shot-firing, and when the blue marl had been reached the innermost freezing-pipes were removed. Cast-iron tubbing, 15¾ feet in inside diameter, was now put down in two lengths of 73 and 29 feet respectively, having 10 segments and 140 screwbolts to each 5 feet ring. The thickness of the upper portion of the tubbing was 1·33 inches of the lower portion 1·66 inches.

After the completion of this tubbing, instead of the strata being allowed to thaw the sinking was continued through grey and blue chalk marl down to 172 feet from bank, whilst the freezing solution was made to circulate as before, through the outermost pipes which had not been removed. From this chalk marl water flowed into the shaft at the rate of about 3 cubic feet per minute, but the waters from the overlying strata were completely shut off. The cast-iron tubbing being completed down to the 172 feet level, the frozen strata were thawed by leading steam through the pipes, and these were then taken out. The sinking thereafter progressed without any noteworthy occurrence, and, at the time of the author's visit, No. 10 shaft was in coal-measures at a depth of 570 feet; the inflow of water at this depth was only 1 cubic foot per minute.

The Poetsch process was made use of from the first in sinking No. 10 bis shaft; 54 days were occupied in the preliminary borings, etc., and the process of freezing was continued for 75 days. The sinking was carried on by means of blasting with compressed black powder; the gunpowder fumes were taken off by an air-channel which communicated with a chimney on the surface. The structural arrangements were, on the whole, much the same as in No. 10 shaft: cast-iron tubbing, 12 feet in internal diameter, and wooden wedging-cribs were used. In the middle of October, 1892, the shaft was yet being deepened; it was lighted by electric (incandescent) lamps, and six-hour shifts of eight men each were busied in building up the tubbing. The temperature in the shaft varied between 23 and 25 degs. Fahr., and it was not the intention of the engineers to thaw the frozen strata until the shaft had been carried down to a depth of at least 170 feet from the surface.

No. 10 bis shaft was completed in 229 days, or an average advance of 0·66 foot per diem, a result which, under the circumstances (water-feeder of 850 cubic feet per minute) was marvellous.

In November, 1892, another shaft was being sunk by the Poetsch process in the Pas de Calais, at the Dourges collieries.

The drowning of No. 10 shaft up to sea-level had the result of providing the shaft with sufficient resistance to withstand the crushing-in pressure of the rocks (whilst arrangements were being made to restart the sinking by the Poetsch process), and the author suggests that in similar cases which may arise in the future, it would be advantageous to flood a shaft by artificial means. Further, he draws attention to the following conclusions:—

1. Hard, dense rocks offer no obstacle to the freezing process. The much-jointed loose chalk was frozen into so firm a mass that blasting was found necessary to break it down.
2. Neither walls nor iron tubbing suffer from the frost.

3. The chances of ill-success with the Poetsch process are largely obviated if, as in the case of No. 10 and No. 10 bis shafts, the freezing process is only put an end to after the provision of a sufficiently high foundation beneath the ice wall or mass of frozen strata.
4. Clayey bands alternating with sandy or stony beds constitute a distinct disadvantage, for the clay freezes much more slowly and freezes less hard than the other strata. Herein practice confirms the experiments of Mr. Alby.* Indeed, plastic clay on freezing assumes a shaly structure, and in cases where it alternates with sands, the dissimilar behaviour of the strata induces breakage of the freezing-pipes. Breakages could perhaps be avoided if the lining-tubes of the preliminary boreholes were left in place instead of being withdrawn, so that the freezing-pipes within them would be throughout in direct contact with a sheet of ice. Another disadvantage of alternating sands and clays or other strata is the production of fissures in the ice-wall, but it is fair to state that in the two sinkings above described such fissures have had no injurious effect worth mentioning.
5. Mr. Reumaux stated that an additional advantage of the Poetsch process consisted in its small cost, but sufficiently full data in support of this assertion are not, so far, forthcoming. O. S. E.

COAL-SCREENING IN THE UNITED STATES.

The Iron Breaker at Drifton, with a Description of some of the Machinery used for Handling and Preparing Coal at the Cross Creek Collieries. By ECKLEY B. COXE. *Transactions of the American Institute of Mining Engineers*, 1890, *vol. xix., pages* 398-474, *and* 43 *plates.*

Anthracite coal as it comes from the mines is not marketable, owing to the fact that it will not burn freely unless the lumps are practically of a uniform size. In addition, a considerable quantity of shale occurs with the pure coal. Slate coal is lumps composed partly of slate (shale) and partly of coal, the latter in large masses. On breaking these, pieces of pure coal of marketable size can be obtained economically. Bone coal consists of shale and coal so interstratified that they cannot be separated economically by mechanical preparation.

The coal coming from the mines is first tipped on to a fixed bar-screen, and the large coal passed by shoots on to a movable bar-screen; the small coal falling through both screens is collected together for further treatment. Up to this stage only two sizes are made, each of which is treated separately. The large coal is divided into three sorts—(*a*) shale and slate, which goes to the dirt-heap; (*b*) pure coal, sold as lumps if there is any market for it; (*c*) slate coal, too impure to go to market in its existing condition. Sometimes the shale can be chipped off with a pick, but more generally the mixture has to be crushed by rolls and the product sized by screens, the dirt being removed by hand-picking from the larger qualities and by washing from the smaller ones. The pure coal is also passed through rolls and afterwards sized by screens.

The small coal that passes through the fixed and movable bar-screens is conveyed to two screens, each of which makes three sizes, called "steamboat," "broken," and "egg." The small coal passes to another pair of screens, known as the stove or wet screens. The steamboat coal from both screens passes *via* a picking-shoot to a loading-shoot, provided all this quality can be sold; if it cannot, a portion is passed through a set of rolls and separated by screens into "broken,"

* *Annales des Mines*, series 8, vol. xi., page 56.

"egg," "stove," "chestnut," "pea," buckwheat No. 1, No. 2, and No. 3, and dust. All the coal which passes to the stove or wet screens, is there divided into seven of the sizes above-mentioned, ranging from "stove" to dust, or rather slime, as these screens are worked wet, *i.e.*, a large amount of water is put on them. All the wet coal from these screens is cleaned in jigging coal-washers of the Lührig type.

The rolls employed for breaking the coal differ from those generally adopted, inasmuch as the teeth are continuous from one end to the other. There are no pointed teeth, as it is found that these have a tendency to split up a lump of coal into small pieces in the same way that the blow of a pick does. It has also been found advisable to use a separate set of rolls to break each size, as if a piece of any size is simply broken as nearly as possible into two (for the next size), the amount of small coal made is much less than if the same size were struck near the centre with a pick and broken into a number of fragments. The production of "dust" is avoided as much as possible, as it is quite unsaleable.

The movable bar-screens are a modification of the Briart screen. Each screen consists of a series of double bars placed sufficiently far apart to allow coal of the required size to pass through. Each set of bars is driven by an eccentric, and one set is always below the second set when moving forward, but above when moving backward. In the Briart screen the shafts which carry the bars rise and fall the same distance as they move forward, while in this construction they only move up and down half as much as they move forward. It was found that with the Briart construction the coal was thrown up and down too much when it was fed forward with any rapidity.

Perhaps the most interesting feature is the movable screen designed by Mr. Coxe, where the screening surface is approximately horizontal, but the motion and action is similar to that which a moulder gives to a sieve when screening sand. Although such screens have been used in small sizes in metal-mining, the chief difficulty in constructing a large gyrating screen making a number of sizes was to support it so that it would gyrate easily and safely, and at the same time be self-contained, so that the centrifugal force will be counterbalanced and will not shake the building. This has been done by a method which essentially consists in supporting one horizontal plate upon another by means of three or more double cones, while the motion of gyration is given to the upper plate by a crank upon a shaft passing through and journalled in the lower plate. Some of these double gyrating-screens have 4 to 8 shelves put in, make from 5 to 9 sizes, and weigh 10 tons: the screening-surfaces have always circular holes, varying from $5\frac{1}{4}$ inches to $\frac{1}{16}$th inch in diameter. The inclination of the shelves is slight, varying from $\frac{1}{8}$th inch to the foot (for the largest size) to $1\frac{1}{2}$ inches to the foot (for the dust.)

The automatic slate-pickers which are used depend for their action on the fact that while the coal generally breaks into cubical masses, the pieces of shale of the same length and width are of much less thickness. Hence, if a quantity of coal and shale has been passed through a screen and properly sized, the shale, if placed edgewise, would drop through a slit over which the coal would pass. The shale-pickers consist of a series of V-troughs, one side of the V being shorter and at right angles to the other. The lower half of the casting has a taper slit in the short side, and these slits widen as they approach the lower end. This is an important point, as if the slits were made parallel they would soon clog.

The height of the breaker from the railway track to the point where the coal is tipped is 79 feet, and the greatest amount of coal which has been screened and cleaned is a little over 260 tons per hour, but the breaker has not been running long enough to determine its maximum capacity. No costs are given. H. W. H.

MINERS' CHANGING AND WASH-HOUSES.

Beschreibung einiger Wohlfahrtseinrichtungen auf der Königlichen Steinkohlengrube Dudweiler bei Saarbrücken. *By* — FABIAN. *Zeitschrift für das Berg-, Hütten-, und Salinen-Wesen im Preussischen Staate*, 1892, *vol. xl.*, *pages* 493-501.

All the buildings are situated close to the shaft, and are so arranged that the miners are only exposed to the outside air for a short time. There is a large waiting-room, which communicates on one side with the lamp-room and on the other with a refreshment bar, where coffee and bread can be obtained at moderate prices. Connected with this room is the changing and bath-room, 108 by 64 feet, and 16½ feet high. The baths are cells fitted with a warm-water douche, and the room contains in all 55 cells, each capable of holding two men. The walls of the cells are made of corrugated iron, and there is a wooden partition in front of the baths, shutting them off from the rest of the room. The room is also fitted with a few cold-water douche-baths. Between two rows of bath cells a railing is fixed to which ropes passing over pulleys near the ceiling are attached. There is a hook at one end of the rope on which the miner may hang his clothes and then pull them up to the top of the building. Each man has his own particular hook which is numbered. It is found that the clothes dry more quickly when pulled up into the warm air near the ceiling.

The water is warmed by steam to a temperature of 95 degs. Fahr., and if the temperature sinks below this, the fact is notified by the ringing of an automatic electric-bell. Each bath requires about 7 gallons of water. The time allowed for the use of a bath for two men is five minutes. The present arrangements allow 1,200 workmen to bathe themselves in one hour. Out of 2,340 workmen 865 use the bath-house regularly.

The cost of the bath-house was £2,144, or £39 per bath or cell.

Special mine-waggons are used for men meeting with an accident. These waggons are fitted with springs and cushions, and are made so as to go into the cage.

The following articles are kept in a house close to the shaft for use in case of an explosion:—(1) One small machine-ventilator on wheels, and fitted with carrying bands; (2) 100 yards of zinc air-pipes, with a few bends, hanging wire, and cloth for plugging holes; (3) rolls of brattice-cloth; (4) saws, picks, hammers, and nails; (5) three small hand-ventilators; (6) 100 yards of small pipe for the hand-ventilators; (7) water-bottles and straps; (8) vinegar for filling the water-bottles; (9) notebooks and pencils; (10) torch-lamps for surface lighting; (11) small pipes, screws, tools, etc., for ventilating and pumping requirements; (12) a portable fire-engine with hose.

W. F. W.

SULPHUR ON PIT-HEAPS.

Sur des échantillons de soufre provenant de la houillère du Perron (Ougrée.) *By* A. COCHETEUX. *Annales de la Société Géologique de Belgique*, 1885-86, *vol. xiii.*, *Bulletin*, *pages* 140-141.

The author, noticing a strong odour of sulphurous acid gas proceeding from a pit-heap at Perron, examined the heap, and found, just below the surface, sulphur varying from lemon-yellow to orange in colour, and forming long needles and spherical agglomerations. The materials composing the heap were pyritous shales from the Coal-measures, and ashes and clinker from the boiler-grates. The sulphur had been liberated probably by reaction taking place between the oxygen of the air, the sulphides, the organic matter, the water, and the carbon dioxide present.

O. S. E.

THE SULPHUR-MINES OF ALTAVILLA-IRPINA, ITALY.

Zur Geologie von Unteritalien, Das Schwefelbergwerk von Altavilla-Irpina in Unteritalien. *By* W. DEECKE. *Neues Jahrbuch für Mineralogie, etc.*, 1891, *vol. ii., pages* 39-48.

The Altavilla-Irpina mines are situated in the valley of the Sabato, about equidistant from Avellino and Benevento (7 miles) and are worked by a company. A new junction-railway between Avellino and Benevento will shortly be opened, and the line passes close by the works. Boys and girls are employed for the lighter duties in the refinery; work is carried on for nine months out of the twelve.

The strata in the Sabato valley are of Pliocene age; they are bounded by the Nummulitic limestones of Petruro and those of Monte Vergine. The sulphur occurs in a greasy bituminous clay, overlain by very thick conglomerates, with which are intercalated sandstones and more or less sandy marls.

The sulphur-deposits are of great thickness, and the beds beneath them have not yet been struck; the sulphur occurs in ramifying veins in the clay, or in lumps or nodules amidst the abundant lenticular masses of gypsum. As the clay becomes more bituminous the sulphur-contents diminish in quantity. Fragments of stems and branches, now converted into lignite, occur, but no other fossils have been found. It is presumed that the sulphur has been separated out from the gypsum by the slow decomposition of the organic matter present in the clay. O. S. E.

MACHINE FOR SHAPING MINING TIMBER.

Note sur la Machine à Façonner les Bois des charbonnages de Sars-Longchamps et Bouvy. *By* ALFRED MATHIEU. *Revue Universelle des Mines, etc.*, 1892, *vol. xviii., pages* 50-52, *and* 5 *figures.*

The machine described is a circular saw with the addition of a special tool for shaping the ends of the timbers. This tool is carried on the same shaft as the saw. The machine is worked by a belt from a steam-engine and makes 1,200 revolutions per minute. One workman can shape 3,000 pieces per day, as much as eight men could do if the shaping was done in the mine, besides which all the waste-material remains on the surface and can be utilized. W. N. A.

TELETHERMOMETERS.

Zwei neue Fernthermometer. *By* HANS HARTL. *Zeitschrift des Vereines Deutscher Ingenieure*, 1891, *vol. xxxv., pages* 1399-1401.

Two instruments are described, intended to indicate temperatures at a distance. The principle of their construction is that the saturated vapours of ether and alcohol respectively or of ether-*cum*-alcohol, increase much more rapidly in tension with increasing temperature than does atmospheric air.

One of the instruments, consisting of a V-shaped glass tube, contains mercury in its two branches; on to the surface of the liquid metal in one branch a few drops of ether are brought, and there is an alarum connected with the instrument, indicating to the observers placed in another room when the previously-fixed limiting temperatures have been reached, between which the temperature of a given space is sought to be maintained.

The second instrument is provided with a constant indicator in the shape of a Wheatstone bridge-arrangement. O. S. E.

CENTRIFUGAL VENTILATORS.

Centrifugal Ventilators. By R. VAN A. NORRIS. *Transactions of the American Institute of Mining Engineers,* 1891, *vol. xx., pages* 637–677.

The following Table of Fan Experiments is recorded:—

Number of Fan.	Experiment.	Owner and Location.	Type of Fan.	Diameter.	Width.	Number of Inlets.	Diameter of Inlets.	Shutter?	Revolutions per Minute.		Cubic Feet of Air per Minute.	Water-gauge.	Horse-power in Air.	Indicated Horse-power of Engine.	Efficiency of Engine and Fan.	Remarks.
									Engine.	Fan.						
				Ft. In.	Ft. In.		Ft. In.					In.				
1	A	Susquehanna Coal Company, Double fan	Guibal	20 0	6 0	4	8 10	Yes	42	84	236,684	1·80	67·13	88·40	75·9	Two fans on separate shafts, driven by a pair of 20 in. by 36 in. engines, belted to drive 2 to 1, with two 24 in. 8-ply rubber belts. Each fan has two inlets and a sheet-iron disc.
	B	Do. do.	Double	..	..	..	..	..	50	100	336,862	2·50	132·70	155·43	85·4	
	C	Do. do.	..	..	..	..	..	..	50	100	298,080	2·60	121·94	143·42	85·02	
	D	Do. do.	..	..	..	..	..	..	55½	111	347,396	3·20	175·17	209·64	83·6	
	E	Do. do.	..	..	..	..	..	..	61½	123	394,100	3·60	223·56	295·21	75·7	
	F	Do. do.	..	Only left-hand fan running					50	100	188,888	1·40	41·67	97·99	42·5	
	G	Do. do.	..						65	130	274,876	2·00	85·63	194·95	44·6	
2	A	Susquehanna Coal Company, No. 6 slope	Guibal	20 0	6 0	2	8 10	No	59	59	59,587	1·20	11·27	16·76	67·83	Same as No. 1 fan, except tips of blades radial. No shutter, no disc. Pair 16 in. by 24 in. engines.
	B	Do. do.	..	..	..	..	..	..	83	83	82,969	2·15	27·86	48·54	57·38	
3	A	Susquehanna Coal Company, Ross seam, No. 2 shaft	Guibal	25 0	8 0	1	11 6	Yes	40	40	49,611	0·87	6·80	13·82	49·2	Single opening, 24 in. by 32 in. engine.
	B	Do. do.	..	..	..	..	..	..	70	70	137,760	2·55	55·35	67·44	82·07	
4	A	Susquehanna Coal Company, Lee seam	Guibal	25 0	8 0	1	11 6	Yes	62	62	225,288	1·45	51·44	62·93	81·74	Same as No. 2 fan. Single inlet. Blades flat. 18 in. by 36 in. engines.
5	A	Delaware and Hudson Canal Company, Baltimore Tunnel	Guibal	17 6	4 0	4	8 0	No	..	28	68,337	0·10	1·07	..	..	Fans run with throttle full open, and 75 per cent. of boiler pressure assumed as mean effective pressure. Two fans on same shaft. Pair 16 in. by 30 in. engines, 24 in. belt.
	B	Do. do.	Double	..	..	..	..	..	..	50	147,232	0·50	11·60	28·55	40·63	
	C	Do. do.	..	..	..	..	..	..	..	69	205,761	1·00	32·42	45·98	70·50	
	D	Do. do.	..	..	..	..	..	..	..	96	299,600	2·15	101·50	120·64	84·10	
6	A	Union Coal Company, Pennsylvania Colliery	?	18 0	6 4	2	8 0	No	80	80	66,560	1·35	14·13	23·60	60·	14 in. by 24 in. engine.
	B	Do. do.	..	..	..	..	..	..	100	100	92,163	2·10	30·50	43·37	70·	
	C	Do. do.	..	..	..	..	..	..	120	120	109,440	2·90	50·00	81·03	61·	
7	A	Mineral Railroad and Mining Company, Cameron Colliery	Guibal	18 0	4 11	2	8 0	No	56	110	71,590	2·45	18·00	30·80	59·	18 in. by 36 in. engine.

CENTRIFUGAL VENTILATORS.—*Continued.*

Number of Fan.	Experiment.	Owner and Location.	Type of Fan.	Diameter.	Width.	Number of Inlets.	Diameter of Inlets.	Shutter?	Revolutions per Minute.		Cubic Feet of Air per Minute.	Water-gauge.	Horse-power in Air.	Indicated Horse-power of Engine.	Efficiency of Engine and Fan.	Remarks.
									Engine.	Fan.						
				Ft. In.	Ft. In.		Ft. In.					In.				
8	A	Lehigh and Western Baltimore Coal Company, Stanton old fan	Guibal	35 0	11 8	1	17 6	No	46	46	228,850	1·95	70·62	98·45	71·73	20 in. by 48 in. engine.
9	A	Lehigh and Western Baltimore Coal Company, Stanton new fan	Guibal	35 0	11 8	1	17 6	No	46	46	234,830	1·85	68·46	94·60	72·37	20 in. by 48 in. engine.
	B	Do. do.	..	..	..	..	..	..	55	55	277,380	2·75	92·93	159·02	57·81	
10	A	Lehigh and Western Baltimore Coal Company, Empire	Guibal	24 0	10 0	1	12 0	No	66	66	134,662	1·95	41·38	62·70	66·00	17 in. by 36 in. engine.
	B	Do. do.	..	..	..	..	..	..	73	73	194,200	2·50	75·69	101·07	75·69	
11	A	Lehigh and Western Baltimore Coal Company, S. Wilkes-Barre	Capell	12 6	12 0	2	7 4	No	50	150	177,280	1·35	37·70	90·10	41·80	24 in. by 36 in. engine.
	B	Do. do.	..	..	..	..	..	..	60	180	215,200	1·65	56·00	123·00	45·30	
11½	A	Lehigh and Western Baltimore Coal Company, Hollenback	Guibal	35 0	11 8	1	17 6	No	45	45	217,338	2·25	77·05	84·4	91·30	20 in. by 48 in. engine. This efficiency is doubtful.
	B	Do. do.	..	..	..	..	..	..	50	50	257,520	2·40	97·40	131·3	74·20	
12	A	East Howle Colliery, Durham, England	Capell	12 0	10 0	2	7 0	No	50	200	133,198	3·35	70·30	102·79	68·40	Pair engines, 20 in. by 36 in. One 24 in. belt. Driving 4 to 1.
	B	Do. do.	..	..	..	..	..	..	50	200	156,510	3·30	81·35	113·61	71·60	
	C	Do. do.	..	..	..	..	..	..	50	200	180,809	3·05	86·89	129·07	67·30	
	D	Do. do.	..	..	..	..	..	..	50	200	209,150	2·80	92·50	150·08	61·70	
13	A	South-West Coal and Coke Company	Guibal	25 0	8 0	1	1 0	No	10	10	28,896	0·10	0·45	1·30	35·	20 in. by 36 in. engine.
	B	Do. do.	..	..	..	..	..	..	20	20	57,120	0·20	1·80	3·70	9·	
13½	C	Morewood Mine	..	..	..	..	..	..	25	25	66,640	0·29	2·90	6·10	48·	
	D	Do.	..	..	..	..	..	..	30	30	73,080	0·40	4·60	9·70	47·	
	E	Do.	..	..	..	..	..	..	35	35	94,080	0·50	7·40	15·00	48·	
	F	Do.	..	..	..	..	..	..	40	40	112,000	0·70	12·30	24·90	49·	
	G	Do.	..	..	..	..	..	..	50	50	132,700	0·00	18·80	38·80	48·	
	H	Do.	..	..	..	..	..	..	60	60	173,600	1·35	36·90	66·40	55·	
	I	Do.	..	..	..	..	..	..	70	70	203,280	1·80	57·70	107·10	54·	
	J	Do.	..	..	..	..	..	..	80	80	222,320	2·25	78·80	152·60	52·	

CENTRIFUGAL VENTILATORS.—*Continued.*

Number of Fan.	Experiment.	Owner and Location.	Type of Fan.	Diameter.	Width.	Number of Inlets.	Diameter of Inlets.	Shutter?	Revolutions per Minute.		Cubic Feet of Air per Minute.	Water-gauge.	Horse-power in Air.	Indicated Horse-power of Engine.	Efficiency of Engine and Fan.	Remarks.
									Engine.	Fan.						
				Ft. In.	Ft. In.		Ft. In.					In.				
14	A	Lehigh Valley Coal Company, Dorrance Colliery	Guibal	35 0	12 0	1	17 6	No	48	48	177,000	2·25	62·70	82·20	76·2	24 in. by 42 in. engine.
15	A	Lehigh Valley Coal Company, Henry Colliery	Guibal	30 0	10 0	1	15 0	No	45	45	245,000	1·35	52·12	60·22	86·55	30 in. by 48 in. engine.
16	A	Lehigh Valley Coal Company, Packer No. 5	?	20 0	6 0	1	12 0	No	58	58	84,149	0·95	12·60	23·04	54·25	17 in. by 24 in. engine.
	B	Do. do.	..	..	..	..	..	..	80	80	117,573	1·80	33·35	54·60	61·00	
17	A	Pleasley Colliery, Lancashire, England	Guibal	24 0	8 0	..	..	Yes	..	70	85,285	2·44	32·74	40·00	80·18	Fan fitted with Walker shutter, 20 in. by 20 in. engine.
	B	Do. do.	..	..	..	..	..	..	..	80	98,512	3·00	46·56	61·12	76·17	
	C	Do. do.	..	..	..	..	..	..	..	90	107,390	3·65	61·71	82·12	75·14	
18	A	Delaware, Lackawanna, and Western Railroad, Avondale	Open	12 0	3 4	2	6 0	..	12½	26	17,640	0·08	0·23	0·63	36·80	14 in. by 30 in. engine.
	B	Do. do.	..	..	..	..	..	..	25	52	29,498	0·25	1·16	2·33	49·87	
	C	Do. do.	..	..	..	..	..	..	50	104	51,842	0·70	5·72	12·36	46·26	
19	A	Delaware, Lackawanna, and Western Railroad Company, Bellevue	Open	16 0	4 6	2	8 0	..	57	110·6	132,282	1·80	38·09	58·92	64·00	The air measurements in all the Delaware, Lackawanna, and Western Railroad Company's experiments were made with the anemometer held stationary in centre of airway, giving quantities of air much too high. Shutter. Engine 16 in. by 30 in.
	B	Do. do.	..	..	..	..	..	..	59	114·5	160,714	1·60	40·57	58·71	69·00	
	C	Do. do.	..	..	..	..	..	..	57	110·6	188,146	1·20	35·57	58·00	61·40	
	D	Do. do.	..	..	..	..	..	..	58	112·6	152,188	1·90	45·56	68·15	66·8	
	E	Do. do.	Closed	..	..	..	..	Yes	58	112·6	142,159	1·60	35·84	51·52	69·5	
	F	Do. do.	Closed	..	..	..	..	No	58	112·6	145,885	1·75	40·22	54·22	74·2	
	G	Do. do.	Open	..	..	..	..	..	..	112	None	1·35	..	..	0	
	H	Do. do.	Closed	..	..	..	..	Yes	..	112	,,	2·30	..	..	0	
	I	Do. do.	Closed	..	..	..	..	No	..	112	,,	2·57	..	..	0	
	J	Do. do.	Open	..	..	..	..	..	57	110·6	136,387	1·20	25·78	..	..	
	K	Do. do.	Closed	..	..	..	..	No	57	110·6	154,738	1·60	39·01	..	..	
	L	Do. do.	Closed	..	..	..	..	Yes	57	110·6	143,597	1·50	33·94	..	..	

CENTRIFUGAL VENTILATORS.—*Continued.*

Number of Fan.	Experiment.	Owner and Location.	Type of Fan.	Diameter.	Width.	Number of Inlets.	Diameter of Inlets.	Shutter?	Revolutions per Minute.		Cubic Feet of Air per Minute.	Water-gauge.	Horse-power in Air.	Indicated Horse-power of Engine.	Efficiency of Engine and Fan.	Remarks.
									Engine.	Fan.						
				Ft. In.	Ft. In.		Ft. In.					In.				
20	A	Delaware, Lackawanna, and Western Railroad Company, Holden fan	Guibal	25 0	8 0	2	12 6	No	43	43	135,600	0·70	14·96	23·95	62·5	18 in. by 36 in. engine.
	B	Do. do.	..	..	..	..	..	..	43	43	113,140	0·80	14·23	22·85	62·4	Mine choked.
	C	Do. do.	..	..	..	..	..	..	40	40	194,510	0·47	14·56	20·71	70·3	Mine doors open.
21	A	Delaware, Lackawanna, and Western Railroad Company, Dodge fan	Open	16 0	4 6	2	8 0	..	48	88·6	177,855	0·85	23·82	43·97	54·2	Mine normal. 18 in. by 36 in. engine.
	B	Do. do.	Guibal	..	..	..	..	..	60	110·8	201,135	1·10	34·86	70·49	49·5	Mine normal. 18 in. by 36 in. engine.
	C	Do. do.	..	..	..	..	..	..	48	88·6	231,225	0·75	27·33	44·93	60·8	Mine doors open.
22	A	Coxe Brothers and Co., No. 2 slope, Drifton	..	20 0	6 0	1	7 0	Yes	62	62	51,084	1·00	8·05	19·82	40·62	Engine 16 in. by 24 in.
23	A	Philadelphia and Reading Coal and Iron Company, Henry Clay Colliery	Guibal	21 0	7 2	1	10 6	No	70	70	126,000	0·85	16·8	73·00	23·	Engine 14 in. by 36 in. Air measurements taken in same manner as for fans 19, 20, and 21.
24	A	Philadelphia and Reading Coal and Iron Company, Mahanoy City Colliery	Guibal	15 0	5 2	1	7 6	No	100	100	65,000	1·60	16·4	68·00	24·1	Engine 12 in. by 30 in.
25	A	Philadelphia and Reading Coal and Iron Company, W. Bear Ridge Colliery	Guibal	18 0	6 2	1	9 0	No	80	80	81,000	2·75	34·67	58·00	59·8	Engine 16 in. by 18 in.

M. W. B.

THE EDUCATION OF MINING ENGINEERS.

By J. H. MERIVALE.

The need of a thorough training for mining engineers has been recognized upon the Continent for very many years. The first mining school established was, the author believes, that of Schemnitz, in Hungary, founded in 1760. This was followed by the establishment of similar schools at Freiberg, Paris, and at most, if not all, of the more important mining centres of the Continent. It was not, however, until 1851 that the first mining school—the Royal School of Mines, Jermyn Street—was established in England; and this good step has been tardily followed, until at last, more than one hundred years after our Continental rivals, we have science colleges, each with its mining department, fairly well distributed throughout the country. Whether these colleges are liberally supported or not, both with money and students, it is not the purpose of this paper to discuss.

The author has been struck with the ignorance of many persons, capable of devoting the necessary time and money to a thorough training in mining: ignorance both of the kind of education required, and of the opportunities now offered for obtaining it; and he has been fortified in this view by a request from the secretary (Mr. M. Walton Brown) for a paper upon the education of mining engineers. He has thought therefore that a schedule of the courses of study provided for mining students at some of the most important of the mining schools of the world, prefaced by a short paper embodying his own views upon this important subject, might be acceptable to the members of the Institute.

The schedule of courses has involved much labour, and he wishes here to thank Mr. Ritson, himself an old student of the Clausthal school, for his very kind assistance; and the secretary of the Institute, who, if he imposed this task upon him (the author) has ably assisted him in carrying it out.

The mining student should receive the usual education of an English gentleman, and should pass the examinations in general education

required by all the professions, excepting that of the engineer. This is not only expedient, but necessary, as the universities require it for their science degrees. He would then be about sixteen years of age, and should leave school and go through a two years' course of pure science at one of the science colleges, taking a diploma at the termination of the course. At the same time he should at least keep up his French, German, and drawing, though he will not have the time to make much advance in these important subjects.

At eighteen years of age he should be old enough to definitely decide upon his profession. If he determines after all not to become a mining engineer, the education he has received will be equally good for almost any other profession. It is a mistake to differentiate too early, and the author has observed this mistake to be made over and over again.

Having decided to be a mining engineer the student must then select a colliery for his five years' apprenticeship. This is an important matter; the colliery should be one where varied work may be seen and, what is still more important, where the viewer has had some experience in training youths, and is willing to take some trouble with them.

Trailing round the pit behind an overman is not putting the time to the best advantage. Each department should be taken in turn, and the youth should himself do the work, so far as he can. Let him be put in charge of an intelligent deputy-overman, take the regular deputy-overman's shift, and really assist in the work, and the same with rolleywaymen, shifters, cashier, storekeeper, etc. If he is strong and is careful not to overdo it, he will be the better for a few weeks' coal-hewing.

He should try to get a little reading, and if he has a taste for science should devote a few hours regularly each week to this, so as to take his B.Sc. Great facilities are offered at the Durham College of Science in the shape of evening lectures, etc., to apprentices desirous of combining a little science with their practical work.

At the end of three years the student should have obtained an accurate practical knowledge of mining in all its details, as practised under one or two sets of conditions. He should now try to get wider views; and this he can best do by going abroad for twelve months, spending six months, say, in France or Belgium, and six months in Germany. He should select mines where the conditions are dissimilar from those to which he has been accustomed, and must be careful not to visit too many pits—some half-a-dozen—*i.e.*, two months at each will be quite enough. He should take copious notes, and neither speak, read, nor write a word of English during his stay abroad.

On his return to England he must, in accordance with the requirements of the examining board, hold some official position for twelve months, that of back-overman is perhaps the best, giving as it does great opportunities of gaining practical experience. He will now be twenty-three years of age and should sit for his certificate, which he will have no difficulty in obtaining.

For the first few years of his professional life, the young colliery manager will probably not have a very large practice, and he may wisely devote some time to further study : take his M.Sc., or even the D.Sc., and write papers for the mining institutes.

In conclusion, the author would wish to point out that though a man must live, money is very far from everything, and each one in youth should devote some portion of his time to the acquirement of those arts without which life for most of us would not be worth having. The mining engineer who neither smokes, photographs, nor fiddles, nor has cultivated any art outside his profession, is only half a man, though he be a D.Sc. of Durham University, and have the biggest practice in the kingdom. And the author is happy to say that some at least of these extra professional pursuits are cultivated in the Durham College of Science.

APPENDIX.

EDUCATIONAL INSTITUTIONS WHERE COURSES OF STUDY ARE PROVIDED FOR MINING AND METALLURGICAL ENGINEERS.

THE ROYAL COLLEGE OF SCIENCE, LONDON, WITH WHICH IS INCORPORATED THE ROYAL SCHOOL OF MINES.

The Royal College of Science at South Kensington is supported by the State, to supply systematic instruction in the various branches of physical science to students of all classes.

The Royal School of Mines is incorporated with the Royal College of Science. Students entering for the associateship of the School of Mines obtain their general scientific training in the Royal College of Science.

The associateship is granted in certain divisions or lines of study. Students who go through any one of these in the prescribed order, and pass the necessary examinations, receive a certificate of associateship of the Royal College of Science, or of the Royal School of Mines.

The associateship of the Royal School of Mines is given in one of the following divisions :—(1) metallurgy, or (2) mining.

The course of instruction, which lasts for three years, is the same for all the divisions during the first and second years (after which it is specialized) in accordance with the following scheme :—

First Year.—Chemistry, mathematics, physics, astronomical physics, and free-hand drawing.

Second Year.—Mechanics, machine drawing, elementary geology and mineralogy. Instruction is given in mathematics so far as may be necessary, and in descriptive geometry.

Metallurgy.

Third Year.—Metallurgy, assaying, and determinative mineralogy.

Mining.

Third Year.—Mining, metallurgy, assaying, determinative mineralogy, and mine surveying.

Examinations are held at the end of each course of instruction, and at such other periods as may be found necessary. On the results of these examinations the successful candidates who have attended regularly and otherwise conformed to the rules of the school are arranged in two classes, first and second.

There are also honours examinations for the subjects of the third year, the successful candidates being placed in order of merit. Honours are only obtainable by candidates for the associateship. No one will be allowed to attempt the honours examination who fails to obtain a first class in the ordinary examination. The candidate for honours will be required to pass an examination in some special branch or branches of one of the subjects in which the associateship can be taken. The papers set will be of a more advanced character than those in the ordinary associateship examination. The candidate will also be required to present a thesis on some special subject, and great weight will be given to the adequate description of any original work or research which has been carried out by the student in one of the laboratories of the school with the sanction of the professor. For the honours examination the subjects will be as follows, and the student will only be required to take up one of these:—

Division I.—Metallurgy.—Researches connected with the practical examination of a metallurgical process or details of a process, and theoretical metallurgy.

Division II.—Mining.—Description of some mine or mining district from personal study, with full details.

A student obtains the associateship who passes in all the subjects of the first two years, and in those of the special division which he selects for his associateship.

The obligation to go through the elementary, or first part in any of the foregoing courses, may be remitted in the case of any student who satisfies the council that he has received sufficient theoretical and practical instruction in those subjects.

The fees for the first two years amount to about £75, and for the remainder of the course for the associateship they vary from £30 to about £40. The private and State-aided students are required to furnish themselves with certain instruments and apparatus before the commencement of the courses.

No student will, except under very special circumstances, be entered for the associateship course unless he has passed in the first class of the elementary stage of mechanics, mathematics, chemistry, and physics, or in some higher stage of those subjects, at the May examinations of the department of Science and Art, or can show to the satisfaction of the council of the college, by having passed the examinations of other recognized examining bodies, that he possesses the necessary elementary knowledge of those subjects.

UNIVERSITY COLLEGE, BRISTOL.

The college supplies instruction in those branches of applied science which are more nearly connected with the arts and manufactures.

Civil Engineers or Surveyors.

The course of civil engineering has been arranged to extend over two years, but special arrangements will be made for students who may deem it advisable to remain for a third year.

First Year.—Physics, engineering, chemistry, mechanics, engineering and geometrical drawing, mathematics, and surveying.

Second Year.—Mathematics, engineering, physics, geology, engineering and geometrical drawing, bridge-construction, and surveying.

The compounded fee for the preceding course is, for each year, 28 guineas, and for second and third years 21 guineas, with a registration fee of £1 1s.

Mining Engineering.

Students who intend to become mining engineers follow the preceding course and special provision is made for their instruction in the third term of each year.

Any student who attends a course of instruction in any subject during a session, and obtains a place in the first or second class in the examination at the end of the course, is entitled to receive a certificate from the college.

An honours certificate is granted, after examination by an external examiner, in conjunction with the professor or lecturer on the subject, to any student of the college who has attended a course of instruction in any one of the subjects.

Associateships.

The title of associate of the University College, Bristol, is conferred on any candidate who (*a*) has attended an amount of instruction in the college equivalent to three courses of three hours per week each in each of three terms during two sessions; and (*b*) has obtained some one of the four following distinctions, namely:—(i.) College honours certificates in four subjects; (ii.) a certificate, granted by the examiners and approved by the council, of having produced an essay or original investigation of exceptional merit, together with honours certificates in three subjects; (iii.) a degree in any university in the United Kingdom; or (iv.) such distinction in the Oxford University examination for women, or the Cambridge University higher local examination, as from time to time shall be considered sufficient.

The title of associate in engineering is granted, on application, to all candidates who have complied with condition (*a*), and who obtain an honours certificate in engineering, and in three of the subjects, mathematics, physics, geology, or chemistry; provided that one at least of their certificates be in mathematics.

CAMBORNE SCHOOL OF MINES, CORNWALL.

The school is worked on the assumption that its students are for the most part engaged in the mines during the day. Instruction in the various scientific and technical subjects is therefore given mainly in the science school in evening classes; and only those subjects, such as assaying, surveying, etc., which demand more time

and are themselves of a practical character, are taught during the day. Facilities for working at the mines are afforded to students by the managers of some of the principal mines in the neighbourhood.

The school course extends over a period of three years, the subjects being arranged as set out below (in the case of a student working for a special course certificate, the arrangement of the subjects can be modified):—

First Year.—Freehand drawing, mathematics, geometry, chemistry, and physics. Machine-drawing, building-construction, or carpentry and joinery may be taken, if geometry is one of the subjects chosen.

Second Year.—Machine-drawing, mineralogy, building-construction, geology, hydrostatics, dynamics, applied mechanics, and mining.

Third Year.—Advanced stages of the foregoing subjects, together with steam, ore-dressing, and metallurgy.

Special Course Certificates.

The special course certificates are of three kinds: mining, mechanical, and chemical. The certificates are of two classes. The qualifications specified below give the minimum required for a pass. To obtain the higher certificate more advanced stages of these subjects count; and such other certificates as the applicant may send in will be considered on their merits. To qualify for a certificate in any course, passes in at least four of the following subjects must be obtained:—Mathematics, geometry, hydrostatics, dynamics, physics, and chemistry. But a certificate of matriculation at any of the universities will be held equivalent.

Mining Course Certificate.

To obtain a mining course certificate, in addition to the preceding, the student must produce a certificate of having worked underground as a miner for at least twelve months; and pass mining (first class advanced stage), ore-dressing (honours stage), and a practical examination in surveying, blowpipe analysis and assaying.

THE DURHAM COLLEGE OF SCIENCE, NEWCASTLE-UPON-TYNE.

This college represents the faculties of science and engineering in the University of Durham, and thus fulfils the functions of a University College in the North of England.

Complete courses of instruction are provided for mining engineers, mechanical engineers, marine engineers, naval architects, electrical engineers, metallurgists, and chemical manufacturers.

Associateship in Physical Science.

Every candidate for the associateship in physical science will be required to satisfy the examiners in mathematics, physics, chemistry, and either geology, natural history, or biology in an examination to be held at the end of the candidate's first year. At the end of the second year, every candidate must pass an examination in the higher parts of two of the subjects, in the elementary parts of which the candidate passed an examination at the end of the first year.

Associates in science are admissible one year after obtaining the title of associate, and having some time before or during their college course passed the examination in general knowledge, to examination for the degree of bachelor of science of the University of Durham.

Mechanical, Mining, or Civil Engineer.

Associates in physical science, who, after their admission to this rank, have been engaged for three years at least in some practical work of mechanical, mining, or civil engineering approved by the council, and have subsequently passed a further examination, having, principally, reference to the work in which they have been engaged, shall be admissible, by grace, to the title of mechanical, mining, or civil engineer of the University of Durham.

The following courses of study are recommended :—

Associateship in Science.

First Year.—Mathematics, physics, practical physics, chemistry, practical chemistry, and either geology, biology, or natural history.

Second Year.—Any two of the following subjects :—Mathematics, physics, chemistry, geology, biology, or agriculture.

Metallurgy.

First Year.—Mathematics, physics, chemistry, mechanical drawing, German or French, and physical and chemical laboratories.

Second Year.—Chemistry, metallurgy, geology, mechanical drawing, German or French, and chemical and metallurgical laboratories.

Third Year.—Metallurgy, chemistry, geology, engineering, and chemical and metallurgical laboratories.

Mining.

First Year.—Mathematics, physics, chemistry, geology, mechanical drawing, and chemical and physical laboratories.

Second Year.—Mathematics, mechanical engineering, mechanical drawing, surveying, coal and metalliferous mining, and technical electricity.

Third Year.—Students able to devote a third year to their studies should attend a special course on chemistry, mechanical and electrical engineering, and the engineering laboratory.

Degrees in Engineering.

Students who intend to proceed to a degree in engineering are recommended to pursue the following course of study. The degree of B.Sc. in engineering may be taken in any one of the four following branches of engineering :—(*a*) Mechanical engineering and naval architecture ; (*b*) electrical engineering (including mechanical engineering) ; (*c*) mining ; and (*d*) civil engineering.

Mining (including Mechanical Engineering).

The course of study (extending over three years) recommended for those who intend to proceed to the degree of B.Sc. in mining is :—

First Year.—Mathematics, physics, chemistry, geology, biology, natural history, and physical and chemical laboratories.

Second Year.—Mathematics, physics (including technical electricity), engineering, mechanical drawing, and physical and engineering laboratories.

Third Year.—Mathematics, physics (including technical electricity), mining, mechanical drawing, and physical, mining, and engineering laboratories.

This course may be extended over four years in cases where the student is unable to undertake it in three years.

The titles and degrees of the University of Durham in science and engineering are open to evening students who pass the matriculation examination, attend

regularly for not less than ten hours per week (in the evenings or on Saturdays), and pass the same examinations as the day students. Evening students may present themselves for the first examination for the associateship in science at the end of their second session, and for the final examination for associateship at the end of their third session, if they obtain a first class in the first examination, otherwise they may present themselves at the end of their fourth session. Evening students continuing to attend the classes for ten hours weekly may present themselves for the examination for B.Sc. one year after obtaining the associateship.

SHEFFIELD TECHNICAL SCHOOL, SHEFFIELD.

The work of the institution is divided into three departments:—(1) The junior day department. (2) The senior day department:—(*a*) Engineering section; and (*b*) metallurgical section. (3) The evening department consisting, amongst others, of special sections of engineering, metallurgy, mining, and building.

Mechanical Engineering.

The course of study is designed to meet the requirements of mechanical, electrical, or mining engineers, architects, or others engaged in industries where a knowledge of mechanical principles is of importance. The complete course extends over three years, and includes attendance at lectures and classes, experimental work in the laboratories, and practical work in the workshops and drawing office.

The prescribed subjects for the complete course are given below:—

First Year.—Mathematics, applied mechanics, engineering laboratory, physics, machine drawing and geometry, engineering, and workshops. The composition fee is £16 16s.

Second Year.—Mathematics, mechanics, physics, machine drawing and design, engineering, engineering laboratories, and workshops. The composition fee is £17 17s.

Third Year.—The following general course of study may be varied according to the special requirements of each individual student. It consists of mathematics, iron and steel, fuel, machine drawing and design, engineering and electrical laboratories, and workshops. The composition fee is £18 18s.

Mining Engineering.

Students of mining engineering take the subjects of the mechanical engineering course, together with special courses in chemistry and geology, for the first year, and mine surveying, the theory and practice of coal-mining, and practical fuel analysis, for the second year.

Metallurgical Engineering.

The prescribed subjects are given below:—

First Year.—Mathematics, physics, chemistry, iron and steel, machine-drawing, and metallurgical laboratory. The composition fee is £18 18s.

Second Year.—Fuel and refractory materials, mathematics, physics, chemistry, machine-drawing, and metallurgical laboratory. The composition fee is £18 18s.

Third Year.—Geology, mineralogy, mathematics, applied mechanics, machine-drawing, and metallurgical laboratory. The composition fee is £18 18s.

Associateship of the Sheffield Technical School.

The associateship of the school is awarded in engineering or metallurgy to each student who attends the lectures and laboratory work prescribed for the complete three years' day courses, and passes the annual examinations satisfactorily in each of the subjects.

THE YORKSHIRE COLLEGE, LEEDS.

The Yorkshire College is intended to promote the education of persons of both sexes, and in particular to provide instruction in such sciences and arts as are applicable to the manufacturing, mining, engineering, and agricultural industries of the county of York, and in ancient and modern languages, history and literature, medicine, surgery, law, logic, moral philosophy, and any other subjects of university or college teaching.

The Yorkshire College is one of the colleges of the Victoria University, and has power to confer degrees on persons who have pursued a regular course of study, and who submit themselves for examination.

Coal-mining (maintained by the Drapers' Company of the City of London).

The coal-mining course occupies two years, and includes courses of lectures on mining engineering, surveying, chemistry, and geology.

First Year.—The lectures include the chemistry and geology of coal-mining, theory and practice of coal-mining, mining engineering, and colliery management.

Second Year.—The lectures comprise the theory and practice of coal-mining, mining engineering, colliery management, and underground surveying.

SYDNEY TECHNICAL COLLEGE, ULTIMO, NEW SOUTH WALES.

The technical education branch of the department of public instruction, New South Wales, comprises about forty technical schools situated at various centres, and the technical college at Ultimo.

The following are the departments of instruction:—Agriculture, architecture, art, chemistry, commerce, domestic economy, industrial and decorative arts, geology, mineralogy, mining, mathematics, mechanical engineering, sanitary engineering, electrical engineering, pharmacy, physics, sheep and wool training, printing, and classes in tailors' cutting, scientific dresscutting, dressmaking, and ambulance surgery.

Mining and Mining Engineering.

This department provides separate courses of lectures and laboratory instruction in each of the following branches of mining engineering:—Coal-mining, metalliferous mining, mine-surveying, and mining machinery.

For a diploma in either branch of mining engineering, students must attend all the classes and pass the prescribed examinations.

The course for the diploma is as follows:—Coal-mining or metal-mining, mine-surveying, mining engineering, geology and mineralogy, and workshops.

Certificates are awarded to students who attend the classes and pass the required examination at the end of the course.

Diplomas will be awarded to students who have attended a complete course of instruction in any department of science or art, and have satisfactorily passed the prescribed examinations. Students who pass their final examination with the highest order of merit receive a diploma in honours.

A diploma in honours entitles students to the distinction of associate of the Sydney Technical College.

The fellowship of the Sydney Technical College will be conferred upon those who, having obtained the associateship, shall have spent not less than six years in actual practice and shall have done some original and valuable research work, or have contributed to the advancement of the industry in which they are engaged.

UNIVERSITY OF OTAGO, DUNEDIN, NEW ZEALAND.

The university contains faculties of arts, science, medicine, and law; and a school of mines.

School of Mines.

The school of mines was established in 1878. It comprises classes in mining, mining geology, mineralogy, and petrography, mathematics, theoretical and applied mechanics, physics, palæontology, theoretical and technological chemistry, qualitative and quantitative analysis, metallurgy, assaying, general geology, and surveying.

There are four divisions—mining, metallurgical, geological, and mine-surveying. In the first three divisions the course of study can be completed in three years, and in the fourth division in two years.

Students who pass the examinations in any of the first three divisions obtain the distinction, with diploma, of associate of the University School of Mines, Otago. On passing successfully the examinations in the fourth division they are entitled to receive the certificate of mining surveyor.

The following are the courses of study prescribed for the respective divisions:—

Associateship.—Mining Division.

First Year.—Mathematics, general geology, mining geology, theoretical chemistry and chemical technology, applied mechanics, mine and land-surveying, and drawing.

Second Year.—Theoretical mechanics, physics, mineralogy, use of the blowpipe and determinative mineralogy, mining, applied mechanics, mine and land-surveying, drawing, and ore-dressing.

Third Year.—Physics, petrography, mining, mechanical gold-extraction, chemical and metallurgical laboratories, assaying and metallurgy, and drawing.

Associateship.—Metallurgical Division.

First Year.—Mathematics, general geology, mining geology, theoretical chemistry and chemical technology, applied mechanics, and drawing.

Second Year.—Theoretical mechanics, physics, mineralogy, use of the blowpipe and determinative mineralogy, metallurgy (including ore-dressing), applied mechanics, drawing, and chemical laboratory.

Third Year.—Physics, metallurgy, mechanical extraction of gold, assaying and metallurgy, drawing, and physical and metallurgical laboratories.

Associateship.—Geological Division.

First Year.—Mathematics, general geology, mining geology, theoretical chemistry and chemical technology, mine and land-surveying, and drawing.

Second Year.—Physics, mineralogy, use of the blowpipe and determinative mineralogy, mine and land-surveying, surveying practice, biology, drawing, and biological laboratory.

Third Year.—Theoretical mechanics, physics, petrography, palæontology, drawing, geological field practice, and physical and chemical laboratories.

Mining Surveyor's Course and Certificate.

First Year.—Mathematics, general geology, mining geology, theoretical chemistry and chemical technology, mine and land-surveying, and drawing.

Second Year.—Theoretical mechanics, physics, mineralogy, use of the blowpipe and determinative mineralogy, mine and land-surveying, surveying practice, drawing, and physical laboratory.

UNIVERSITY OF KING'S COLLEGE, WINDSOR, NOVA SCOTIA.

The King's College University confers degrees in arts, divinity, law, engineering and science.

The aggregate outlay necessary, for the three years' residence and study qualifying for a degree, may be estimated at about £30 to £40 per year. The mere money payment demanded need never, however, exclude a candidate really prepared to benefit by the course of instruction offered, inasmuch as many benefactors of the university possess the right of nomination. Each nominee is exempt from the payment of certain yearly fees besides the fee for a B.A. degree. There being some forty certificates conferring this privilege, it is easy for students to obtain it. If a scholarship is held besides, nearly the whole cost of education will be covered.

There are at present in the university five schools open to matriculated students of arts, divinity, engineering, science, and civil law.

The course of engineering is framed with the view of giving the student a sound and thorough training in mathematics, pure and applied, a comprehensive knowledge of engineering and applied science, together with such practical experience in work connected with the profession of a civil engineer as it is within the scope of the university to afford.

The course for B.Sc. has been established for carrying on advanced study in mining and chemistry.

Degree of Bachelor of Engineering.

Undergraduates presenting themselves for examination for the degree of bachelor of engineering shall be required to have been duly matriculated in the school of engineering; to have subsequently studied in King's College the course herein prescribed, and to have passed the first university examination for the said degree.

The course for the degree in civil or mining engineering shall usually extend over four years, and shall comprise attendance on the following curriculum:—

First Year.—Mathematics, chemistry, French or German, English literature, surveying and levelling, and field and office work.

Second Year.—Mathematics, analytical chemistry, chemical physics, botany and zoology, French or German, English literature, mensuration, surveying and levelling, office work, and engineering excursions.

Third Year.—Mathematics, mathematical physics, geology, mineralogy (including blowpipe analysis), civil engineering, applied mechanics, geometrical drawing, engineering office and field-work, and geological excursions.

Fourth Year.—Natural philosophy, mathematical physics, civil engineering, and engineering office and field work.

Degree of Master of Engineering.

Every candidate for the degree of master of engineering must have taken the degree of bachelor of engineering, and must have been engaged in practical work connected with the profession of a civil engineer for at least three years from the time of so doing. He must have held a position of responsibility in such connexion.

He will be examined in each of the following groups of subjects:—(1) Practical engineering: the preparation of designs, specifications and estimates for civil engineering work. (2) Applied science: (*a*) applied mathematics; (*b*) technological chemistry, assaying and analysis; (*c*) geology and mining; (*d*) practical electricity. Candidates must select one of the four subjects in the second group. The particular department of engineering work in which the candidate is examined will be chosen with reference to the character of the work in which he has been specially engaged, and in this portion of the examination candidates will be tested as regards

their knowledge of the subject by being required to submit a complete set of drawings, specifications, and estimates for some proposed work, for the preparation of which a month or five weeks will be allowed. These drawings, with the detailed calculations upon which they are based, will be sent in to the examiners, and if they are approved an oral examination of the candidates will be conducted for the purpose of verifying the authenticity of the work submitted, and of testing the candidate's professional ability.

Degree of Bachelor of Science.

The course of study comprises :—

First Year.—Mathematics, chemistry, English literature, French or German, surveying and levelling (with field and office-work).

Second Year.—Mathematics, chemistry, qualitative analysis, chemical physics, botany, zoology, English literature, French or German, mensuration, surveying, and levelling (with office-work).

Third Year.—Mathematical physics (mechanics and hydrostatics), geology (including petrography and field geology), mineralogy (including crystallography and blowpipe analysis), mining, economics, geometrical drawing, French or German, and geological excursions.

Fourth Year.—Mathematical physics (optics and astronomy), economic geology and mineralogy, chemistry (quantitative analysis of minerals, ores, etc.), metallurgy and assaying, geological drawing, economics, and French or German.

Degree of Master of Science.

Candidates for this degree must be B.Sc.'s of three years' standing. Every candidate will be required to send in a thesis on some scientific subject embodying original work, and also, in addition, be required to pass an examination on one of three or more subjects to be named in the calendar :—Coal, iron, and copper (in each case with analysis and methods of mining).

Degree of Doctor of Science.

Candidates for D.Sc. must be B.Sc.'s of eight years' standing or M.Sc.'s of five years' standing. A thesis on some approved subject to be sent to the board of examiners, and a proof of continued successful work in science.

BALLARAT SCHOOL OF MINES, INDUSTRIES AND SCIENCE, UNIVERSITY OF MELBOURNE, BALLARAT, GRENVILLE COUNTY, VICTORIA.

This school was established for the purpose of imparting instruction in the various branches of science relating to mining, the theory and practice thereof included.

There are six courses of study, viz. :—Mining, metallurgy, geology, agriculture, electrical engineering, and civil engineering. In each of these the associateship of the School of Mines can be obtained.

Mining Engineering Course.

First Year.—Mathematics, natural philosophy (mechanics and heat), land surveying, mechanical drawing, mineralogy, geology, and theoretical chemistry.

Second Year.—Mathematics, natural philosophy (sound, light, electricity and magnetism), land-surveying, mechanics (theoretical and applied), mechanical drawing, mineralogy, petrography, geology, analytical chemistry (qualitative), chemical laboratory practice, and metallurgy.

Third Year.—Mathematics, applied electricity, mechanics (theoretical and applied), mine-surveying, mechanics as applied to mining, mechanical drawing, scientific mining, geological surveying, analytical chemistry (quantitative), and assaying.

In order that a student may receive his certificate of associateship in the mining engineering course he must produce satisfactory documentary evidence that he has served for at least one year under a duly qualified mining engineer, mining surveyor, or mine manager.

Metallurgy Course.

First Year.—Natural philosophy (mechanics and heat), mathematics, mechanical drawing, mineralogy, geology, theoretical chemistry, analytical chemistry (qualitative), and chemical laboratory practice.

Second Year.—Natural philosophy (sound, light, electricity and magnetism), mathematics, mechanics (theoretical and applied), mineralogy, petrography, geology, theoretical chemistry, analytical chemistry (quantitative), chemical laboratory practice, metallurgy, and assaying.

Third Year.—Applied electricity, mineralogy, petrography, chemical geology, analytical chemistry (quantitative), chemical laboratory practice, metallurgy, assaying, and mining laboratory-work (ore-dressing, etc.)

Manager's Course.

The curriculum for mine managers is divided into three parts, comprising:—(1) Practical mining, including elementary geology; (2) mine surveying, including plotting and computing; (3) mechanics, as applied to mining, including strength of materials, drainage, ventilation, explosives.

The school's certificate of competency as a mine manager is awarded to each candidate who has passed in all the subjects prescribed. He must also produce satisfactory evidence of having worked underground at least two years in a mine in which not less than fifty men were employed, and in which the operations were carried on by steam-power and machinery. A candidate having had two years' experience as a quartz miner must produce satisfactory evidence of having been employed at least twelve months in an alluvial mine to entitle him to a certificate as a mining manager of alluvial mines.

Intending students are recommended to take a preliminary course of instruction in the following subjects taught in the miners' class, viz.:—Arithmetic, logarithms, elementary trigonometry, mensuration, and Euclid (optional).

SCHOOL OF MINES AND INDUSTRIES, BENDIGO, VICTORIA.

The School of Mines and Industries, Bendigo, was established for the following purposes:—(*a*) To impart sound instruction, chiefly in the various branches of science connected with mining operations, and to instruct students in the theory and practice of mining, in geology, mineralogy, physical geography, meteorology, and in physical, natural, and applied science; (*b*) to thoroughly teach chemistry, metallurgy, and assaying; (*c*) arithmetic, algebra, mathematics, surveying, and

astronomy; (*d*) drawing and painting in their various branches, lithography, wood engraving, and the theory and practice of the mechanical trades, modelling and carving; (*e*) ancient and modern languages; (*f*) shorthand writing, telegraphy, and allied subjects; and (*g*) such other subjects as may from time to time be deemed desirable by the council.

The lectures comprise classes in geology and mineralogy, chemistry and metallurgy, practical mining, engineering and surveying, botany, zoology, architectural and geometrical drawing, machine and building construction, drawing and painting in all its branches, telegraphy, shorthand, the French, German, Latin, and Italian languages, and elocution.

Examinations are held and certificates granted at the end of each course of lectures.

ROYAL SCHOOL OF MINES, PŘZBRAM, BOHEMIA.

The Royal School of Mines embraces a school of mining and a school of metallurgy, either of which can be passed through in one year.

The school of mining comprises lectures on mining, geology of ore-deposits, ore-dressing, mine-surveying, surveying and plotting, theory of mining machines, building-construction, and chemical analysis.

The school of metallurgy comprises lectures on the metallurgy of iron, general metallurgy, manufacture of salt, assaying, assays and analyses, and theory of smelting.

There are also lectures on building-construction, forestry, mining law, book-keeping, metallurgy, mining (in brief), and first aid to injured.

Any ordinary student who has gone through either or both of the two schools can obtain certificates of the results of the examinations.

At the end of each term examinations are held in those subjects the lectures on which have terminated in that term.

HAINAUT SCHOOL OF MINES AND INDUSTRY, MONS, BELGIUM.

The Hainaut School of Mines course extends over four years, at the end of which the diploma of engineer is given to successful students. Candidates of all nationalities are admitted.

The tuition fee is under £5 per annum, but books, apparatus, etc., are not provided by the school.

Diplomas are awarded after courses of instruction in mining, metallurgy, applied chemistry, mechanical engineering, railway engineering, building construction, and electricity.

At the beginning of the fourth year students must decide which of the subjects they intend to take up for their diploma, but it is possible to obtain a diploma in two subjects in the four years. It is impossible to tell from the college programme what is exactly the mining course. It appears, however, that the first two years are devoted to general science and engineering, and the last two years to mining and more advanced portions of the first year's subjects.

SCHOOL OF ARTS, MANUFACTURES, AND MINES, ATTACHED TO THE UNIVERSITY OF LIÉGE, LIÉGE, BELGUIM.

The object of this school is to qualify students for the degree of engineer. It has two divisions, viz., the preparatory and the special school. The preparatory school prepares students for the special school.

The special school is divided into four sections :—Mining, mechanical engineering, electrical engineering, and arts and manufactures. In each of these the degree of engineer can be obtained. The mining division requires a five years' course, including the two years in the preparatory school; each of the other divisions four years, including two years in the preparatory school.

According to Belgian law no one can hold a government appointment as engineer unless he has obtained the degree of engineer. Candidates are examined in the same subjects as those for entrance into the preparatory school, and in the work done during the two years' course in this school. Foreigners may be examined in Latin, or in their native language instead of in French. The fees appear to be about £6 a year.

The course of study in the mining division is as follows :—

First Year.—Grammar, analytical geometry of three dimensions, descriptive geometry, experimental physics, French composition, drawing, and geometrical drawing.

Second Year.—Descriptive geometry, applied and analytical mechanics, elements of astronomy and geodesy, general chemistry, English or German, drawing, and applied geometrical drawing.

Third Year.—Applied mechanics, industrial physics, machine description, mineralogy, assaying, English or German, etc.

Fourth Year.—Geology, working of mines, industrial and inorganic chemistry, metallurgy, industrial architecture, applications of electricity, English or German, etc.

Fifth Year.—Mining, topography, railway working, industrial architecture, metallurgy, industrial geography, industrial economy, law of mines and industries, English or German, etc.

The course also includes visits to mines, works, etc., and geological excursions. There are similar courses leading up to the other engineering degrees. The candidates for degrees may sit for their final examination at the end of each year.

CATHOLIC UNIVERSITY OF LOUVAIN, LOUVAIN, BELGIUM.

The schools, annexed to the Catholic University of Louvain, train engineers for every branch of industry. They grant the following diplomas, viz. :—(*a*) Civil engineer of mines; (*b*) engineer of building-construction; (*c*) engineer of arts and manufactures, and of mines; (*d*) engineer-constructor; (*e*) engineer-architect; and (*f*) conductor of building-construction. (*a*) and (*b*) are obtained after a five years' course; (*c*), (*d*), and (*e*) after a four years' course; and (*f*) after two years. A course consists of at least 200 hours of study in each year.

The course of instruction includes excursions and reports, the students visiting coal-mines of the various Belgian basins, metallurgical establishments, engine-works, etc.

The fees amount to about £10 a year, and in addition there are examination fees varying in amount from £1 5s. to about £3.

THE NATIONAL HIGHER SCHOOL OF MINES, PARIS, FRANCE.

The National School of Mines was founded in the first instance to train engineers for the mining corps. Of late years its doors have been opened to others, both natives and foreigners, anxious to obtain an industrial training. There are no fees for instruction.

There are three classes of students, viz.:—(1) Engineer students. These come from the Polytechnique, and are intended for the service of the State. (2) Day students. These enter the school by competitive examination. (3) Foreign students. These are admitted, after passing a qualifying examination, on the recommendation of the ambassadors or chargés d'affaires of foreign powers. They are not compelled to follow the complete course, but can select at the beginning of the year the line of study which they wish to pursue, and in this only will they be examined, and will receive a certificate. There is also a fourth class of students known as free students. These are either natives or foreigners authorized by the Minister of Public Works to follow the whole or a part of the course. They do not attend the practical work, or the examinations, nor do they receive a diploma.

The length of the special course of study is three years; and in addition there is the preparatory course of one year.

The training at the School of Mines is especially directed to imparting a knowledge of all matters relating to minerals. But, besides this, instruction is given on the general conditions of modern industry, machines, railways, building construction, legislation, and economics.

A course of military art is provided for those students holding a place in the army; and, lastly, German and English are taught, to enable the students to study the scientific works written in those languages.

The courses of instruction are as follows:—

Preparatory Year.—Mechanics, analysis and descriptive geometry, physics, chemistry, drawing, and preparations for the examinations.

Special Course.

First Year.—Working of mines, metallurgy, analytical chemistry, industrial chemistry, mineralogy, palæontology, topography. German and English, drawing, mining and metallurgical designs, mineral analysis, and preparation for the examinations.

Second Year.—Metallurgy, analytical chemistry, geology and petrography, machines and strength of materials, railways, economics, German or English, metallurgical and machine design, mineral analysis, industrial and gelogical excursions, and preparation for the examinations.

Third Year.—Applied geology, building and machine-construction, legislation, applications of electricity, artillery, German or English, and preparation for the examinations.

DOUAI MINE-OVERMEN'S SCHOOL, DOUAI, FRANCE.

The object of this school is to train workmen for the position of overmen, etc. The students are boarded at the school, paying about £20 a year. The length of the course is two years, half of which is spent in a mine. The students are taught grammar, arithmetic, elementary geometry, so much trigonometry and geometry as is required for surveying, physics and chemistry, etc. Diplomas are given at the end of the course.

Candidates must be at least sixteen years old, and have worked in a mine They must also pass an examination in reading, writing, and arithmetic.

SAINT ETIENNE SCHOOL OF MINES, SAINT ETIENNE, LOIRE, FRANCE.

The object of the school is to train engineers, managers of mines, and managers of metallurgical works. The instruction is free, and extends over a period of three years. The first year is devoted to pure mathematics, mechanics, physics, mineral analysis, mineralogy, descriptive geometry, stereotomy, and planning. The two other years to practical work:—Working of mines, washing and dressing of minerals, machines and buildings, metallurgy, mineral analysis, geology, railways, law of mines, vegetable palæontology, and book-keeping. The students visit the neighbouring mines.

Diplomas and certificates are given to those who have satisfactorily gone through the course of study. Foreigners are admitted, after having passed an examination to show that they are capable of following the course of instruction.

Candidates for admission must be Frenchmen or naturalized Frenchmen (though foreigners are also admitted) between 17 and 26 years of age, and must pass an entrance examination. The students are not boarded. The cost of board and lodging is about £1 a week. The cost of the uniform is about £8.

BERLIN ROYAL GEOLOGICAL AND MINING INSTITUTE, BERLIN, GERMANY.

The geological department is for making maps, etc., of the geological formations of the German empire.

The mining department for teaching students the theoretical and practical parts of mining, metallurgy, surveying, etc. The following subjects are taught in the mining department:—Mining, metallurgy, metallurgy of iron, assaying, blowpipe analysis, technical chemistry, land and mine surveying, machine construction, designing chemical works, building construction, mechanical drawing, trigonometry, analytical geometry, differential and integral calculus, mechanics, mineralogy, petrology, geognosy, palæontology, analytical chemistry, mining law, etc.

The whole course takes three years, the latter part of which is spent in practical work. German students are admitted on producing certificates of attendance at any German technical school or college. Students can go up for examination in any subject they choose, and will be given certificates—provided they satisfy the examiners.

ROYAL TECHNICAL COLLEGE, AIX-LA-CHAPELLE, GERMANY.

The Technical College aims at affording education for a technical calling in the Government or civil service and in commercial life.

The Technical College is under the supervision of the Government Minister for Education, and consists of the following departments:—(1) Architecture; (2) civil engineering; (3) mechanical engineering; (4) mining, metallurgy, and chemistry; (5) general sciences, especially for mathematics and natural philosophy.

The lectures are arranged in yearly courses. Only in exceptional cases does a course of lectures extend over part of a year only.

The admission of a German to the college is dependent on his producing a leaving certificate from a German public school or commercial college. A foreigner can be admitted into the college if the warden and director of the department, which he wishes to enter, are satisfied that he is qualified for the same by age, education, etc.

On joining the college each student receives a certificate of admission which is available for four years, but which can be prolonged if required.

Examinations are held in the case of one year lectures, at the end of the college year; and in the case of the half-year lectures, at the end of each half-year. Students may enter for final examination in those subjects, the lectures on which they have attended, and receive certificates for the same.

Students who have followed the four years' course of instruction in any of the departments are entitled to enter for a diploma examination, and receive a diploma testifying to their acquirements and practical knowledge, and entitling them, as the case may be, to the degree of (*a*) architect; (*b*) metallurgical chemist; (*c*) civil engineer; (*d*) metallurgical engineer; (*e*) mechanical engineer; (*f*) surveyor; (*g*) electrical engineer; (*h*) mining engineer. These examinations take place in two parts, the preliminary at the end of the second year, and a principal examination on the completion of the course.

The appointment as an official in the Government Architectural Service is obtained by passing a preliminary and two principal examinations, these, according to the branch which they concern, being divided into :—(*a*) Building; (*b*) civil engineering; (*c*) mechanical engineering.

For the mining department of the Government Service there are special rules and regulations. The course of instruction is as follows :—

Mining, Metallurgy, and Chemistry.

The course includes :—Mineralogy and crystallography, with practical exercises in determination of minerals; petrography, practice in the mineral collection; guidance to independent crystallographical, mineralogical, and petrographical work; experimental inorganic chemistry, chemistry of benzol and pyridin, practical chemistry, general and inorganic experimental chemistry; introduction to metallurgical experiments, mining, ore-dressing, salt-mining, chemistry of the metals, volumetric analysis, historical chemistry; introduction to metallurgy, metallurgy of iron, general metallurgy, iron casting, new methods of extracting metals, electro-metallurgy, etc., designing of smelting-works, metallurgical assaying, blowpipe analysis; designing of mining and ore-dressing works, mining law, mine management; technical chemistry, designing of chemical works, practical technical chemistry; land and mine surveying, drawing, practical land and mine surveying, physical geography, ore-deposits, palæontology, structural geology, practical palæontology, elements of mineralogy and geology.

ROYAL SCHOOL OF MINES, CLAUSTHAL, HARZ, GERMANY.

The college is devoted to teaching mining and metallurgy. Students are of two grades: "bergakademiker" and "hospitanten."

The standard of the qualification for admission as a "bergakademiker," in the case of Germans, is the final certificate of a gymnasium, high school of the first class or one of the Royal Prussian Trade Schools. Foreigners have to give proof by means of certificates of equivalent education.

Students with less previous education can be admitted as "hospitanten," provided that their certificates prove that they have had sufficient education to enable them to fully understand the lectures. At the end of a year "hospitanten" who are qualified, have the opportunity of becoming "bergakademiker" if they can pass an entrance examination in a full course of elementary mathematics.

The course of instruction is divided into the practical preparatory course and the ordinary course of lectures.

The practical preparatory course begins every year in the first week after Easter, and is devoted to mining, ore-dressing, and smelting. The fee for each of the branches is 18s.

Students are examined in any of the single subjects, or in all the mining and smelting subjects, for which degrees are given. Examinations are held at the end of each term and a certificate will be given by the director of the academy to students passing examinations in any of the subjects taught.

Mining or Metallurgical Engineer's Certificate.

Students who wish to obtain a degree of mining or metallurgical engineering must, at the end of the first year, pass an intermediate examination, and can take either or both parts.

The cost of board and lodging is from £4 to £6 per month, and the total cost of fees, board, and lodging, would be from £50 to £70 for the college year.

The following course of lectures in the mining department, for students who have attended a technical college before, is recommended to students, but they are free to take any lectures they choose. The director will always advise students on the above point.

First Year.—Trigonometry, algebra, analytical plane-geometry, differential and integral calculus, descriptive geometry, physics, mechanical drawing, chemistry, palæontology, ambulance lectures, general jurisprudence and mining law, analytical solid geometry, and lectures on the commercial part of mining.

Second Year.—Mineralogy, practical mineralogy, mechanics, mining, surveying. geology, qualitative chemical analysis, blowpipe analysis, theory of combustibles, ore-dressing, mining geology, practical physics, and qualitative chemical analysis.

Third Year.—Machine-construction and designing, assaying, mining law, mineral deposits, geotectonic or structural geology, electricity, building construction, blowpipe analysis, and qualitative analysis.

THE ROYAL SAXON ACADEMY OF MINING, FREIBERG, SAXONY.

This school is intended to prepare students for the professions of mining and metallurgical engineers and surveyors. Training for other professions is not given, neither does the curriculum include general education.

Applicants who are not subjects of the German empire may be exempted from the matriculation examination, if they are provided with certificates admitting them to institutions of the same standing in their own land.

Partial exemption from the matriculation examination may be permitted to applicants of mature years who substitute experience in practical work for deficiencies in theoretical knowledge, and to those who can furnish evidence of efficiency in arts instead of science.

The object of the practical course in mining is the acquirement of such information as is desirable for a course of study in mining engineering.

The work of the practical course at the Freiberg mines must be taken as follows:—1. Surface work for eight or nine weeks in ore-dressing and concentration. 2. The time remaining is to be spent in work underground, and also in visiting under the direction of the professor of mining engineering other mines in the district.

The fees for tuition appear to amount to about £10 a year for natives and £15 a year for foreigners; but the programme is not very distinct upon this point.

The full course in mining extends over a period of four years :—

First Year.—Mathematics, descriptive geometry, spherical trigonometry, physics, inorganic chemistry, mineralogy, crystallography, drawing, and planning.

Second Year.—The higher parts of some of the above subjects, mechanics, mining, geology, palæontology, economic geology, blowpipe analysis, and mechanical drawing.

Third Year.—Surveying, mining, engineering, metallurgy, building construction, and machine-drawing.

Fourth Year.—Surveying, general and mining law, mining and metallurgical buildings, calculations and statistics, political economy, sanitation, and technical electricity.

At the close of the course the student sits for a final examination, and, in the event of passing, obtains a diploma.

HALLE AND ANHALT MINING SCHOOL, EISLEBEN, SAXONY.

The school is divided into two departments, the former being preparatory for the latter. There are four preparatory schools situated in various parts of Saxony, and the higher school is situated at Eisleben.

The object of these schools is to train officials for the mines and works of the Halle district. As, however, foreigners are not admitted it is unnecessary to give here an abstract of the educational course.

STOCKHOLM POLYTECHNIC SCHOOL, STOCKHOLM, SWEDEN.

The Mining College at Stockholm forms a part of the Polytechnic School, and includes courses of either three or four years, of which the last only is entirely devoted to the science of mining.

During the first year instruction is given in mathematics, mechanics (theoretical and practical), the construction of machines, physics (general and applied), chemistry, architecture, geology and mineralogy, geodesy, geometry, constructive drawing and freehand.

The last year is occupied with the study of metallurgy, working of mines and testing of ore and docimacy, and mechanics. The pupils pass great part of the summer in visiting mines and ironworks.

Hitherto the instruction has been gratuitous, but hereafter a small fee will be required. The pupils who have passed through a complete course obtain certificates of capacity.

TOKIO COLLEGE OF ENGINEERING, TOKIO, JAPAN.

The College of Engineering forms one branch of the University of Tokio. The students enter after a six years' course in a preparatory school, where there are from 1,200 to 1,500 boys. There they improve their English, French, and German; learn all mathematics required by engineers, including the calculus, have some forty lectures in geology, and mineralogy, and considerable laboratory practice. Formerly about forty of these, but now about ninety, go every year into the engineering

college, and others to the medical, law, literature, and pure science branches of the university. The geology taught in the engineering college is purely technical.

The mining course should consist of one hundred and twenty lectures, but sickness, typhoons, earthquakes, etc., usually reduce it to about one hundred. One year a typhoon stopped the classes for a week, and last year an earthquake stopped them for nearly three weeks.

All lectures are given in English. Much of the designing work goes into practice, the rest into models for the museums, which are now said to be better than anything of the kind in Europe, America, and Australia. There is a large workshop always running, with a variety of motors (steam, electric, gas, and water), a metallurgical laboratory, and a fair number of dressing-machines.

The summer vacations are spent at mines, and all the students who have passed through the college find employment in Japan, China, Corea, etc.

The mining course extends over three years, and is as follows:—

First Year.—Mining, mineralogy, geology, steam engine, mechanism, building-construction, surveying, determination of minerals, qualitative analysis, designing, and metallurgy.

Second Year.—Underground surveying, metallurgy, ore-dressing, water motors, pumps, cranes, etc., determination of minerals, assaying, blowpipe analysis, and quantitative analysis.

Third Year.—Mining, metallurgy, ore-deposits, mechanical and metallurgical technology, metallurgical experiments, mechanical engineering, mining designing, metallurgical designing, and mining law.

Adjoining the main college building is a dormitory, where the students of the college reside.

UNIVERSITY OF ARIZONA, TUCSON, ARIZONA, UNITED STATES.

The departments of instruction are:—(1) Science, literature, and the arts; (2) teachers' training, and elementary instruction; (3) agriculture; (4) normal department; and (5) mineralogy and the school of mines. The third and fifth departments were established as the beginning of the university. The fourth department is provided for by legislative enactment in the popular and well-managed Territorial Normal School at Tempe. The first and second departments will be established when the income of the university will permit.

The tuition is free. Matriculation fee, to be paid but once, £1. Students are charged for material used in laboratories. A dormitory for the accommodation of a limited number of students has been fitted up in the university building, to be used until the students' dormitory is built, where students will be boarded at cost price, which will not exceed £4 per month. Books cost from £1 to £2 per year.

The School of Mines.

This college of the university subserves a twofold object, firstly, the thorough training of young men in the sciences of mining and metallurgy to such an extent as to fit them to undertake the development of the mineral resources of the country, after a supplementary period devoted to practical work; and, secondly, to make use of the laboratories for tests, experiments, and investigations of practical utility to mining industries.

Mining Engineering.

The course of mining engineering is as follows :—

First Year.—Algebra, English, physics, projection-drawing, drill, rhetoric, physical laboratory work, geometry, descriptive geometry, and lettering.

Second Year.—Geometry, calculus, chemistry, general botany, modern language, drill, trigonometry and land-surveying, zoology, and topographical surveying and field work.

Third Year.—Analytical mechanics, mine-surveying, chemical laboratory work, modern language, resistance of materials, mineralogy, geology, dynamics, electrical engineering, and assaying.

Fourth Year.—Mining engineering, engineering geology, mining law, wind-wheels and hydraulics, practical work, metallurgy, mining accounts, mine administration, mine examination and report, thesis and practice.

Metallurgical Engineering.

The course in metallurgy is identical with the mining course in the first and second years; in the later years the course is as below:—

Third Year.—Architectural drawing and designing, mechanics, modern language, resistance of materials, mill work, mineralogy, geology, dynamics, analysis of ores, geology, assaying, and quantitative chemical laboratory.

Fourth Year.—Metallurgy, analysis of fuels, fluxes, etc., wind-wheels and hydraulic engines, mill accounts, practical mill-work, memoirs, administration, thesis and practice.

UNIVERSITY OF CALIFORNIA, SAN FRANCISCO, UNITED STATES.

The schools of the University of California are situated at various places in the State, and furnish instruction in literature and in science, and in the professions of law, medicine, dentistry, and pharmacy. In the colleges of letters, agriculture, mining, mechanics, civil engineering and chemistry, in the literary course, and in the course in letters and political science, these privileges are offered without charge for tuition, to all persons qualified for admission.

In the professional colleges in San Francisco, except that of law, moderate tuition fees are charged.

Eight regular courses of study are at present established, leading directly, under conditions hereinafter stated, to corresponding degrees, namely :—

Faculty of the college of letters, comprising :—(*a*) Classical course, leading to the degree of A.B.; (*b*) literary course, leading to the degree of B.L.; and (*c*) course in letters and political science, leading to the degree of Ph.B.

Faculties of the five colleges of science, comprising :—(*a*) Agriculture; (*b*) mechanics; (*c*) mining; (*d*) civil engineering; and (*e*) chemistry, each of which leads regularly to the degree of B.Sc

College of Mining.

The college of mining is designed for students who wish to become mining or metallurgical engineers, or to engage in one of the many pursuits connected with mining industry, such as the surveying and mapping of mines, the assaying and extraction of ores, the designing and use of mining machinery, or the working of mines.

The undergraduate course may be completed by the average student in four years.

During the first two years considerable time is devoted to linguistic studies, embracing English prose style, to the preparation of summaries and précis writing, and to the acquisition of a reading knowledge of either French or German. Owing to the large and valuable scientific and technical literatures of these languages, it is important to the advanced student to be able to read them both. Only one is required of undergraduates, but both may be mastered during the course by a little extra effort. Those who have already a reading knowledge of either of these languages are advised to select the other in college. A knowledge of Spanish, while not required, will be of considerable advantage to mining students, as they will find many professional opportunities in Spanish-American countries. It may be easily acquired during the course. Preliminary to the strictly technical studies of the course, the student receives a sufficient training in those branches of modern physical science which lie at the basis of all the industries connected with mining: on the one hand, mathematics and its applications; and on the other, chemistry, mineralogy, petrography, and geology.

The mathematical studies are pursued with special reference to subsequent practical applications in surveying, physics, and analytical mechanics, which in their turn serve as a means for discussing such subjects as strength of materials and hydraulics. A similar sequence is observed with other studies; thus descriptive geometry is connected with work in the drawing-room, surveying with extensive field-practice and the mapping of surveys actually made by the student, physics with physical problems and work in the physical laboratory, strength of materials and hydraulics with original designs in the drawing-room, followed by working tests in the mechanical laboratory. These important studies extend through the whole four years' course; and inasmuch as they are peculiarly interdependent, it is necessary that they should be completed in the prescribed order, otherwise the student will find himself seriously embarrassed in his work.

The studies in chemistry and allied branches begin with general experimental chemistry, inorganic and organic, followed by analytical chemistry as an application, viz.: qualitative, quantitative and blowpipe analysis, subjects indispensable to subsequent work in metallurgy and assaying.

Having acquired a working power in chemistry, the student begins the study of mineralogy; this is followed by petrography, and by courses in general and field geology. These studies are developed with special reference to their applications in mining.

The technical branches of mining, metallurgy and assaying, peculiar to this college, are begun in the junior year, when the student has had sufficient training in the general and preparatory branches to study them with profit.

In the course in metallurgy, after the general consideration of the subjects which concern the treatment of all the metals, the rest of the undergraduate work is devoted to a detailed study of all the important methods in use for the reduction of the ores of lead, silver, gold, copper, and mercury.

The course concludes with a written thesis on some subject connected with mining or metallurgy, and leads to the degree of bachelor of science.

Candidates for the professional degrees in this college must satisfy the following conditions:—

Degree of Mining Engineer.—The candidate must be a graduate of the college of mining of this University, or give satisfactory evidence of having successfully pursued a course of study equivalent to its regular undergraduate course. He must also pass a satisfactory examination in the following subjects:—Mining, ore-dressing, petrography, economic geology, thermodynamics (elements), drawing

and construction of mining machinery, blowpipe assaying, and political economy. He must have had at least one year of actual practice in the field in the course chosen, and must show, by an original memoir upon some subject bearing upon this profession, power to apply his knowledge to practice. This degree will not be given earlier than three years after graduation.

Degree of Metallurgical Engineer.—The candidate must pass an examination in the following subjects:—Metallurgy, ore-dressing, assaying and analysis, blowpipe assaying, thermodynamics (elements), drawing and construction of furnaces and metallurgical machinery, and political economy. In all other respects the conditions are the same as those required for the degree of mining engineer.

COLORADO STATE SCHOOL OF MINES, GOLDEN, COLORADO, UNITED STATES.

The organization of the School of Mines of Colorado resembles that of the best technical schools of the United States. It is a school of applied science, in which, however, more than usual weight is given to those branches having a more or less direct bearing upon mining and metallurgy. All tuition is free.

There are four full courses of study, viz.:—Civil engineering, mining engineering, metallurgy, and electrical engineering. Each covers a period of four years The studies, however, are identical during the first two years of all courses, beginning to diverge at the opening of the third year. The course in mathematics is taken in full by all, except the students in the metallurgical course.

The degrees given are:—(*a*) Civil engineer (C.E.); (*b*) engineer of mines (E.M.); (*c*) metallurgical engineer (M.E.); (*d*) electrical engineer (E.E.); and (*e*) bachelor of science (B.Sc.) The B.Sc. degree is given to any student of four years' residence, who, after having completed the course of the first two years, devotes himself for two more years to the study of analytical and theoretical chemistry. Such students are allowed to attend lectures in the higher classes on other topics than the specialty they have chosen, and must, during their last year, conduct some special research in analytical or technical chemistry.

Students will not be admitted to the fourth year as applicants for the degree of C.E. unless they have shown very marked ability in mathematics. They must also have been connected with some survey or other active field-operation on engineering lines during one of the vacations of their course.

Each student is required, during the summer vacation preceding his senior year, to execute a memoir on some assigned subject. At the end of the third year the student is also assigned a subject for a graduating thesis, such data being given as would be met with in practical experience.

The course of studies is as follows:—

First Year (*for all courses*).—Algebra, geometry, geology, freehand drawing, mechanical drawing, chemistry, and qualitative analysis.

Second Year (*for all courses*).—Plane and spherical trigonometry, descriptive geometry, qualitative and quantitative analysis, crystallography, physics, mechanical drawing, surveying, determinative mineralogy, algebra, shadows and perspective, stoichiometry, physical laboratory, and blowpipe work.

Mining Engineering.

The special courses in mining engineering for third and fourth years are as follow:—

Third Year.—Analytical geometry, differential and integral calculus, civil engineering, primary and secondary batteries, mining, quantitative analysis, mechanical drawing, metallurgy, mechanics, theory of strains, mining engineering, ore-dressing, electrical units, and vacation memoir.

Fourth Year.—Integral calculus, kinematics, mechanical drawing, mechanics, dynamics, theory of strains, economic geology, metallurgy, dynamo-electric machinery, distribution of electricity for lighting, thermodynamics, plans, constructions and estimates, mechanical engineering, electricity in mining, thesis work (including plans, estimates, and drawings), and long-distance transmission of power. In addition to the above schedule, one review study is taken in each term.

Metallurgical Engineering.

The special courses in metallurgy during the third and fourth years are as follows :—

Third Year.—Mining, quantitative analysis, metallurgical chemistry, civil engineering, metallurgy, mechanical drawing, mechanics, civil engineering, ore-buying and smelting-charges, ore-dressing, and a vacation memoir.

Fourth Year.—Kinematics, metallurgy, economic geology, plans, constructions, and estimates, theory of strains, applied chemistry, slag calculations, mechanical drawing, economics of metallurgy, dynamics, thermodynamics, mechanical engineering, and thesis work (including plans, estimates, and drawings). A review study is also taken in each term.

THE UNIVERSITY OF ILLINOIS, URBANA, CHAMPAIGN COUNTY, ILLINOIS, UNITED STATES.

The organization of the university comprises colleges of:—(*a*) Agriculture ; (*b*) engineering, with courses in mechanical engineering, electrical engineering, civil engineering, municipal and sanitary engineering, mining engineering, architecture, and architectural engineering ; (*c*) science ; (*d*) literature ; (*e*) additional schools of military science and of art and design ; together with (*f*) a graduate school.

College of Engineering.

The courses of instruction comprise :—Mathematics—advanced algebra, trigonometry, conic sections, analytical geometry, differential calculus ; theoretical and applied mechanics—analytical mechanics, resistance of materials, hydraulics ; general engineering drawing—elements of draughting, descriptive geometry, lettering ; physics—elementary mechanics and sound, heat and light, electricity and magnetism ; mining engineering—mine attack, mine-surveying, ore-dressing, mine engineering ; civil engineering—land-surveying, topographical drawing and surveying, transit surveying and levelling, railroad-engineering, masonry-construction, geodesy, practical astronomy, bridges.

Labour is furnished as far as possible to all who desire it. It is classified into educational and remunerative labour. Educational labour is designed as a practical instruction, and constitutes a part of the course in several schools. Students are credited with their proficiency in it as in other studies. Nothing is paid for it. Remunerative labour is prosecuted for its products, and students are paid what their work is worth. The usual rate paid for ordinary farm, garden, and shop labour is 6d. per hour. Students of sufficient experience may be allowed to work by the piece or job, and thus by diligence or skill secure more pay.

Some students who have the requisite skill, industry, and economy, pay their entire expenses by their labour; but, in general, young men cannot count upon doing this at first, without a capital to begin with, either of skill, or money to serve them till a degree of skill is acquired.

The tuition is free in all the university classes. The cost of residence varies from £30 to £60.

Mining Engineering.

The course comprises the greater part of the pure and applied mathematics of the courses in mechanical and civil engineering. Much time is devoted to chemistry and geology, with the addition of metallurgy and other technical studies peculiar to mining engineering. Students who are graduated from this course are not supposed to be familiar with all the details of mine management from actual experience, but they will have obtained such a knowledge of the principles underlying all successful practice, and such a familiarity with the science of mining in all its branches, that the art may be acquired with the minimum of practice.

Plans, estimates, drawings, reports, and calculations, based upon data obtained in the student's own experience, are constantly required, and no pains are spared to familiarize each member of the class with the duties and responsibilities of every grade, from miner to manager.

The course of the studies is:—

First Year.—Algebra, elements of draughting, chemistry, French, German, or English, trigonometry, descriptive geometry and lettering, analytical geometry, freehand drawing, and military.

Second Year.—Differential and integral calculus, land-surveying, physics, analytical geometry, topographical drawing and surveying, transit-surveying and levelling, French, German, freehand drawing (optional), and military.

Third Year.—Analytical mechanics, mine attack, mineralogy, resistance of materials, assaying, geology, hydraulics, roofs, chemistry, mine surveying, themes and elocution.

Fourth Year.—Mine engineering, ore-dressing, heat engines, geology, hydraulic engines and wind wheels, chemistry, political economy, roofs, and metallurgy.

Degrees.

The usual bachelor's and master's degrees are conferred upon those who satisfactorily complete the courses of study described in the different colleges. A candidate for a bachelor's degree must pass in the subjects of his chosen course, and must conform to the directions given in connexion with that course in regard to optional subjects.

In all cases an accepted thesis is required for graduation, based upon original research, and must contain at least 2,000 words, or an equivalent in tables, drawings, and illustrations.

The degree of bachelor of science will be given to those who complete a course of study in the college of engineering, of agriculture, or of science. The name of the course will be inserted after the degree.

The master's degrees, M.A., M.L., and M.S., and the equivalent degrees of civil engineer and mechanical engineer, etc., will be given, after 1894, to graduates of this or other similar institutions who have pursued at this university a year of prescribed graduate studies and have passed examinations thereon, or who have pursued as non-residents three years of such study and have passed the required examinations. Studies for a master's degree must be in the general line

of the bachelor's degree already received, and of the degree sought. In all cases an accepted thesis is required and this should be presented at least one month before the close of the collegiate year. It must be based upon original research and must show scholarly acquirements of a high order.

MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON, UNITED STATES.

The Massachusetts Institute of Technology includes a Society of Arts, a Museum of Arts, and a School of Industrial Science.

The School of Industrial Science is devoted to the teaching of science as applied to the various engineering professions—namely, civil, mechanical, mining, electrical, chemical, and sanitary engineering, as well as to architecture, chemistry, metallurgy, physics, and geology. Courses of a less technical nature, designed as a preparation for business callings, and a course in biology preparatory to the professional study of medicine, are also given.

The subjects of study have been arranged in twelve distinct formulated courses, each of four years' duration. For the satisfactory completion of any one of these, the degree of bachelor of science is conferred. Of twelve courses, seven give their students scientific and practical training for the various engineering professions; those in chemistry, physics, biology, and geology, having a larger proportion of pure science, afford preparation either for professional practice, for teaching, or for scientific investigation.

In the first year all the courses are the same, and contain subjects which are considered essential for the more strictly professional studies of the later years. At the end of the year the regular student selects the course which he will pursue during the remaining three years, and his work becomes more specialized thereafter as it progresses.

Within most of these regular courses the student is given a considerable latitude in the selection of the branch of his intended profession to which he will specially devote his energies in the later years of his study. This is accomplished by means of options. Thus in civil engineering he may elect sanitary and hydraulic engineering, geodesy, or an advanced course in railroad engineering and management.

Regular Courses.

First Year.—The studies are common to all regular courses as follow:—Solid geometry, algebra, chemistry, chemical laboratory work, rhetoric and English composition, French or German, mechanical drawing, freehand drawing, plane and spherical trigonometry, political history since 1815, and military drill.

Mining Engineering and Metallurgy.

Second Year.—Analytical geometry, physics, German, American history, descriptive geometry, differential calculus, English literature and composition, mineralogy and blowpipe analysis (silver assay), and optional subjects:—(*a*) Principles of mechanism, use of surveying instruments; (*b*) surveying, topographical drawing; (*c*) mechanism: cotton machinery, machine tools, drawing, physical geography; (*d*) surveying and drawing, physical geography. This may be accompanied by a summer course in practical mining or metallurgy and field-work in mineralogy.

Third Year.—Integral calculus, general statics, physics, strength of materials, kinematics and dynamics, physical laboratory work, German, assaying, mining engineering, geology, and optional subjects:—(*a*) Steam engineering valve-gears, thermodynamics, drawing; (*b*) railroad and highway-engineering; (*c*) steam

engineering, boilers, engineering laboratory work; (*d*) railroad and highway-engineering. This may be followed by a summer course in practical metallurgy or mining.

Fourth Year.—Metallurgy, metallurgy of iron, mining engineering, dynamos, mining laboratory work, memoirs, and optional subjects:—(*a*) Strength of materials, friction, steam engineering and hydraulics, engineering laboratory work; (*b*) strength of materials, theory of elasticity, theory of structures, hydraulics, hydraulic measurements; (*c*) strength and stability of structures, theory of elasticity, technical machinery, engineering laboratory work; (*d*) theory of structures, electric railroads, hydraulic engineering.

Another course of instruction is:—

Second Year.—Analytical geometry, physics, German, American history, theoretical chemistry, differential calculus, English literature and composition, mineralogy and blowpipe analysis, and optional subjects:—(*a*) Descriptive geometry, principles of mechanism, blowpipe (silver assay); (*b*) surveying, topographical drawing, blowpipe assay; (*c*) mechanism: cotton machinery, machine-tools, drawing; (*d*) surveying and drawing, physical geography. This may be accompanied by a summer course in practical mining or metallurgy and field-work in mineralogy.

Third Year.—Integral calculus, general statics, physics: heat, strength of materials, kinematics and dynamics, physical laboratory work, German, assaying, analytical chemistry, and optional subjects:—(*a*) Steam engineering, thermodynamics, valve-gears, drawing; (*b*) mining engineering, geology; (*c*) steam engineering, boilers, engineering laboratory work; (*d*) mining engineering, geology, electricity. This may be followed by a summer course in practical mining or metallurgy.

Fourth Year.—Strength of materials, friction, mining engineering, analytical chemistry, metallurgy, memoirs, metallurgical and chemical laboratories work, and optional subjects:—(*a*) Hydraulics, dynamos, English criticism; (*b*) electricity, political economy and industrial history, commercial law; (*c*) engineering laboratory work, technical machinery, English criticism; (*d*) mining engineering, political economy and industrial history, commercial law.

Summer Schools of Mining and Metallurgy are organized for the study of mines, mills, smelting-works, and geological fields. Since the year 1870 these schools have made studies in Colorado, Michigan, Virginia, Vermont, Pennsylvania, Lake Champlain, New Brunswick, and Nova Scotia.

Graduation.

The degree of bachelor of science, in the course pursued, is given for the satisfactory completion of any regular course of study. To be entitled to a degree, the student must have passed satisfactory examinations in all the prescribed studies and exercises, and, if required, a final or degree examination, embracing all the subjects which particularly relate to his course. He must, moreover, prepare a dissertation on some subject included in his course of study; or an account of some research made by himself; or an original report upon some machine, work of engineering, industrial works, mine, or mineral survey; or an original architectural design, accompanied by an explanatory memoir.

The tuition fee for regular students is £40 per year.

Students may conveniently live in any of the neighbouring cities or towns on the lines of the various railroads, if they prefer to do so. The cost of board and rooms in Boston and the neighbouring cities and towns need not exceed from £1 10s. to £2½ per week.

UNIVERSITY OF MICHIGAN, ANN ARBOR, MICHIGAN, UNITED STATES.

The University of Michigan is a part of the public educational system of the State, and aims at completing the work that is begun in the public schools by furnishing ample facilities for a liberal education in literature, science, and the arts, and for a thorough professional study of medicine, pharmacy, law, and dentistry. The doors of the institution are open to all students, wherever their homes.

The matriculation fee for citizens of Michigan is £2, and, for those who come from any other state or country, £5, is paid but once, and entitles the student to the privileges of permanent membership of the university. In addition to the matriculation fee, every student has to pay an annual fee for incidental expenses. Resident graduates are required to pay the same annual fee as undergraduates.

Students obtain board and lodging with private families for about £1 a week. The annual expenses of students, including clothing and incidentals are, on the average, about £80. The university does not undertake to furnish manual labour to students; yet a few find opportunities in the city for remunerative labour.

The university offers to persons who wish to become professional engineers thorough courses of study extending over about four years.

In civil engineering all the technical branches are under the direct care of those who have had professional experience as well as a full scientific training, and in all particulars the course embodies as close an imitation of the requirements of active labour as the instructors who have the several branches in charge can devise.

In mechanical engineering the course of study includes a wide range of special studies. Prominence is given to the study of steam engineering, and in this branch a large amount of practical work is done. The instruction is arranged to accommodate those who wish to devote their time principally to mechanical engineering proper, to steam engineering, or to marine engineering and naval architecture.

In mining engineering and metallurgy the course of instruction, which is intended to cover about four years of study, includes a part of that provided for students in civil and in mechanical engineering, though more especial attention is paid in the latter part of the course to mineralogy, geology, and chemistry. The instruction in the technical branches is arranged so as to meet the wants, both of those whose purpose it is to confine their professional work more closely to metallurgy, and of those who intend to engage in the practice of mining and metallurgy combined.

In electrical engineering the first three years of the course are nearly the same as in mechanical engineering. Besides the preliminary work in mathematics, language, drawing, and physics, instruction is given to pattern making, metal work, forging, and foundry work; and enough of the study of steam engines and other prime movers is included to meet the needs of the professional electrical engineer.

The courses of instruction comprise mathematics (algebra, trigonometry, analytical geometry, and the elements of differential and integral calculus), French and German, English grammar and composition, physics, chemistry, drawing, practice in geometrical and in mechanical drawing, and in the study of descriptive geometry.

The more technical subjects are taken up in the latter part of the course. Some of these subjects are of equal value to all classes of engineering students, such as analytical and applied mechanics, the strength and resistance of materials, and the metallurgy of the useful metals, especially iron and steel; others are adapted more particularly to the wants of the special students in the several courses.

Upon the completion of a prescribed course of study, amounting to twenty-five full courses, and the presentation of a satisfactory thesis, the student receives the degree of bachelor of science. The diploma given indicates the line of study pursued.

Bachelors of arts, of philosophy, of science, and of letters, of this university, and graduates of any other reputable college, are recommended for the same degree with the regular students, after attendance on, and a satisfactory examination in, the technical subjects alone of the several courses. These studies can be completed in two years.

Mining Engineering.

To obtain the recommendation of the faculty for the degree of bachelor of science for a course in mining engineering, the student must satisfy one of the two following sets of requirements:—Mining (French and German, English, mathematics, physics, chemistry, analytical chemistry, mineralogy, geology, drawing, surveying, civil engineering, mechanical engineerimg, mining engineering, metallurgy); or Metallurgy (French and German, English, mathematics, physics, general chemistry, analytical chemistry, mineralogy, geology, drawing, mechanical engineering, mining engineering, and metallurgy).

The Degree of Civil Engineer, Mechanical Engineer, Mining Engineer, and Electrical Engineer.

The degree of civil engineer may be conferred upon bachelors of science of this university who have taken the degree for a course in civil engineering if they furnish satisfactory evidence that they have pursued further technical studies for at least one year, and, in addition, have been engaged in professional work, in positions of responsibility, for another year. The first of the above requirements may be satisfied by pursuing at the university, under the direction of the faculty, a prescribed course of study for an amount of time, not necessarily consecutive, equivalent to a college year. If the candidate does not reside at the university, his course of study must be approved in advance by the professor of civil engineering, and he must prepare a satisfactory thesis on some engineering topic, to be presented, together with a detailed account of his professional work, one month, at least, before the date of the annual commencement at which he expects to receive the degree. The conditions on which the degrees of mechanical engineer, mining engineer, and electrical engineer, as second degrees, are conferred upon bachelors of science of this university who have taken the degree for a course in mechanical engineering, in mining engineering, or in electrical engineering, are analogous in character to those enumerated for the degree of civil engineer.

Doctors' Degrees.

Doctors' degrees are conferred only on persons who have previously received a bachelor's degree, either here or at some other reputable university or college, and also during residence here have made special proficiency in some one branch of study, and good attainments in two other branches, and have presented a thesis that shall evince the power of research and of independent investigation. It is not intended that the doctors' degrees shall be won merely by faithful and industrious work for a prescribed time in some assigned course of study, and no definite term of required residence can be specified; but it is the practice of the university to require at least one full year of residence of candidates that have already earned a master's degree, and at least two full years of candidates that have previously taken only a bachelor's degree.

The degree of doctor of philosophy is open to persons that have received the degree of bachelor of arts, or of bachelor of philosophy; the degree of doctor of science to persons that have received the degree of bachelor of science; and the degree of doctor of letters to persons who have received the degree of bachelor of letters.

THE MICHIGAN MINING SCHOOL, HOUGHTON, MICHIGAN, UNITED STATES.

Mining only is taught at this school. It is situate in Houghton county, in the Portage lake copper-mine district, and within easy reach of the Keweenawan and Ontonagan copper districts, and the iron-mining regions of Marquette, Menominee, and Gogebic.

The course of instruction for the regular students extends over a period of three years, effecting a saving of time in years by continuing the work through most of the year, and by making the course strictly professional and technical:—

First Year.—Mathematics, drawing, physics, chemistry, surveying, mineralogy and lithology, physical, chemical and mineralogical laboratories.

Second Year.—Differential and integral calculus, drawing, properties of materials, chemistry, qualitative and quantitative analysis, metallurgy, petrography, mechanics, applied mechanics, mechanism, electricity, mining, mine-surveying, stratigraphical geology and palæontology, machine shop, and chemical laboratory.

Third Year.—Graphical statics, quantitative analysis, metallurgy, ore-dressing, mechanics of materials, mechanical engineering, electrical engineering, physical geology, metallurgical analysis, assaying, practical geology, mining engineering and mine accounts, hydraulic and structural engineering, economic geology, and chemical laboratory. The student may also take engineering design and electric motors and their applications, or chemical analytical methods. Every student who is pursuing the regular course is required to select either engineering design and electrical engineering, or technical chemistry.

In order to be able to give more time to the technical parts of the education of a mining engineer, it is proposed to lengthen the course of study to four years.

Every student completing the three years' course is required to present to the faculty a satisfactory thesis, embodying the results of an investigation upon some subject related to the studies of that course, before he can be recommended to receive his degree. Students intending to graduate in any year are required to select their subjects and present them to the faculty for approval by the 1st of March of that year.

Under the act of organization the board of control have made the school entirely free, no charge being made for tuition or incidentals, whether the student be a resident of the State of Michigan or of the United States or not; all has been made free to the students from every land.

Arrangements are made whereby those who desire to do so can obtain board and rooms in private families, and in boarding-houses, in Houghton and Hancock, at prices varying from £3 to £5 per calendar month. The necessary expenses of the student may be estimated at about £50 to £90 per annum.

Degrees.

Students who complete the present course of three years, and present a satisfactory thesis, will receive the degree of mining engineer or engineer of mines

(E.M.). The thesis must conform to the requirements detailed above, and must be completed and approved by the faculty before that body will recommend that the degree of mining engineer or engineer of mines, be conferred upon the student.

Those who wish to become candidates for a higher degree will enter under the following terms:—Students who are graduates of this institution, or of others of similar grade, whose course shall be approved by the faculty, will be admitted as candidates for the degree of doctor of philosophy. In order to attain this degree they must pursue for at least two years an advanced course of study in subjects germane to the undergraduate course in this institution, which course of study is to be approved by the faculty. One of the years of study may, in special cases, be spent elsewhere, and the work accepted, on sufficient proof of its thoroughness and high character, as the equivalent of one year's work spent here. But under no condition will the degree be given unless one year at least is spent as a resident worker at this institution.

Students who are both graduates of this or of an equivalent professional school, and also of some college or university whose course of study is accepted by the faculty, may be admitted to the degree of doctor of philosophy, after taking for at least one year an approved course of study at this institution. The degree of doctor of philosophy will only be given in case the student shall have shown marked ability, power for original investigation, has passed a satisfactory oral public examination, and presented a thesis embodying the result of original investigation, which has been approved by the faculty.

Students who are graduates of this or of other institutions having a satisfactory equivalent course, and who shall have pursued here, according to the above regulations, a successful course of study for the degree of doctor of philosophy, may at the same time receive the degree of mining engineer, if that degree has not been conferred at their previous graduation.

THE UNIVERSITY OF MINNESOTA, MINNEAPOLIS, UNITED STATES.

The University of Minnesota is a State institution, being a part of the State educational system. It is situated in the city of Minneapolis, about a mile below and in full view of the Falls of St. Anthony.

The university is composed of the following colleges:—(*a*) Science, literature and arts ; (*b*) engineering, metallurgy and mechanical arts ; (*c*) agriculture, etc.

The university has no dormitories, except for the school of agriculture, but students find no difficulty in obtaining board among the people of the city. Good board can be obtained in private families at prices ranging from £1 upwards per week.

The university cannot promise employment to those desiring to earn their own living. The public bounty stops at furnishing free instruction. Many of the students support themselves while in college, and a young man who really wants work, and will look for it, can generally find it.

The average necessary annual expenses of students boarding in families appear to be about £60, those of students boarding in clubs about £40.

The College of Engineering, Metallurgy, and the Mechanical Arts.

In this college there are seven regular courses of study, viz., civil engineering, mechanical engineering, electrical engineering, architecture, mining, chemistry, and metallurgy leading to the corresponding bachelor's degrees.

Mining Engineers.

The courses of study in mining are as follows :—

First Year.—Algebra, German or French, chemistry, chemical laboratory work, drawing, trigonometry, qualitative analysis, carpentry, surveying, and military drill.

Second Year.—Analytical geometry, French or German, topography, draughting, physics, mineralogy, differential and integral calculus, descriptive geometry, quantitative chemical analysis, and quantitative blowpipe assaying.

Third Year.—Mechanics, geology, quantitative chemistry, assaying, mining, metallurgy, mechanical laboratory work, machine design, lithology, ore-testing and dressing, drilling and blasting, applied geology, and technical essay.

Fourth Year.—Methods and machinery for extracting minerals and ores, metallurgy, geology, electrical engineering, law (optional), prospecting, exploration of mines, etc., hydraulics, steam engines and motors, designs and specifications, mine engineering, and thesis (optional).

Metallurgical Engineering.

The courses of study in metallurgy are :—

First Year.—Algebra, German or French, chemistry, chemical laboratory work, drawing, trigonometry, qualitative analysis, carpentry, surveying, and military drill.

Second Year.—Analytical geometry, French or German, topography, draughting, physics, mineralogy, differential and integral calculus, descriptive geometry, quantitative chemical analysis, and quantitative blowpipe analysis.

Third Year.—Mechanics, geology, quantitative chemistry, assaying, mining, metallurgy, mechanical laboratory work, machine design, lithology, ore-testing and dressing, applied geology, and technical essay.

Fourth Year.—Mining, metallurgy, geology, electrical engineering, law (optional), hydraulics, steam engines and motors, designs and specifications, and thesis (optional).

UNIVERSITY OF MISSOURI, ROLLA, MISSOURI, UNITED STATES.

The School of Mines and Metallurgy is an institute of technology, and constitutes one of the colleges which, taken together, form the University of the State of Missouri.

The course of instruction at this school deals in detail with the principles and the practice of engineering, with special reference to mining engineering, civil engineering, mechanical engineering, chemistry and metallurgy, mathematics, physics and electricity.

At the close of the year each member of the senior class presents a paper in which he records some independent investigation, in a subject congenial to his tastes, and included in the scope of his course.

Provision is now made for the following technical courses:—(*a*) Mining engineering; (*b*) civil engineering; (*c*) mechanical engineering; (*d*) chemistry and metallurgy; and (*e*) mathematics and physics. For the satisfactory completion of any one of these courses the degree of bachelor of science will be given.

The requisites for admission to any of these courses are passing grades in the subjects taught in the preparatory course. The first year is the same for the engineering courses; the selection being made at the beginning of the second year.

In addition to the regular courses outlined above, provision is made for special courses in assaying, surveying, and electricity. For the satisfactory completion of these subjects a certificate of proficiency will be given. The requisite for admission to one of these courses is a knowledge of the preparatory studies of that subject.

Preparatory Course.

The course of study is :—

First Year.—Higher arithmetic, English grammar, composition and rhetoric, algebra, United States history, physiology and hygiene.

Second Year.—English, algebra, physics, solid and plane geometry, and chemistry.

A matriculation fee of £2 is payable on entrance, and a library fee of 10s. a term, payable on the first day of each term, is required of every student. The cost of books and stationery may be assumed to average £2 during the session. Board, including fuel, lights, washing, etc., can be obtained for £2 to £3 a month. The necessary expenses for the school year are from £25 to £40.

Engineering Courses.

The higher courses of study are as follows :—

First Year.—General chemistry, elementary mechanics, descriptive geometry, stereotomy, trigonometry, analytical geometry, chemical laboratory work, field-work, and drawing.

Mining Engineering.

Second Year.—Analytical geometry and calculus, assaying, physics, chemical technology, mineralogy, geology, civil engineering, mining engineering, chemical laboratory work, and physical laboratory work.

Third Year.—Analytical mechanics, metallurgy, electric transmission of energy, dynamo-electric machinery, mining engineering, physical laboratory work, chemical laboratory work, and thesis.

Degrees.

A diploma of graduation is conferred on one who has passed all examinations in any of the following departments :—Mathematics, physics, analytical chemistry, engineering, and the academic course.

The degree of bachelor of science in mathematics and physics is conferred upon one who has passed examination on all of the subjects of instruction in the course of mathematics and physics.

The degree of bachelor of science in civil engineering is conferred on one who has passed examination on all of the subjects of instruction in the civil engineering course.

The degree of bachelor of science in mining engineering is conferred on one who has passed examination on all of the subjects of instruction in the mining engineering course.

The degree of bachelor of science in mechanical engineering is conferred on one who has passed examination on all of the subjects of the mechanical engineering course.

The degree of civil engineer, mining engineer, or mechanical engineer is conferred on one who, having graduated in civil, mining, or mechanical engineering, and having received the bachelor's degree therein, has identified himself with the profession during a period of not less than three years, and during that time has demonstrated by work his fitness for his chosen profession.

WASHINGTON UNIVERSITY, ST. LOUIS, MISSOURI, UNITED STATES.

Washington University, founded in the city of St. Louis under an Act of Incorporation by the State of Missouri, approved February 22nd, 1853, is intended to embrace the whole range of university studies, except theology, and to afford opportunity of complete preparation for every sphere of practical and scientific life.

The Polytechnic School.

The courses of study in the Polytechnic School are seven in number, viz. :—(*a*) Civil engineering; (*b*) mechanical engineering; (*c*) electrical engineering; (*d*) chemistry; (*e*) mining and metallurgy; (*f*) science and literature; and (*g*) special studies and investigations in pure and applied science.

The tuition fee in the undergraduate department is £30 a year. The yearly expenses may range, according to tastes and habits of economy, from £75 to £100.

The studies are the same for all the courses during the first and second years, but during the later years they diverge widely.

First Year.—Higher algebra, pneumatics, heat, laboratory work, French or German, mechanical drawing, English composition, orthographic projections, use of joiner's tools, ethics, plane trigonometry, optics, freehand drawing, descriptive geometry, chemistry, history of England, elocution and composition, and practical wood turning and pattern making.

Second Year.—Analytical geometry, electricity, magnetism, and laboratory work; use of surveying instruments; land, topographical, and mining surveying; field practice, descriptive geometry, sketching, brush shading, topographical and isometric drawing, practical forging iron and steel, differential calculus, hydrographic and railway surveying, curves and turnouts, earth-work, graphical statics in mechanics, qualitative chemical analysis, acoustics and laboratory work, French or German, and English.

Every student is required, during the vacations following the first and second years, to prepare reports upon suitable engineering methods or constructions from personal examinations and studies.

Mining and Metallurgy.

Third Year.—Integral calculus and applications, zoology, analytical statics, dynamics, qualitative chemical analysis (lectures and laboratory practice), qualitative blowpipe analysis, crystallography, mineralogy, botany, assaying, metallurgy, drawing (sections, crystals, plans and sections of mines and mining machinery, furnaces, apparatus and machinery of smelting-works, etc.), practical hand and machine-tool work on metals, sampling, hand-panning, fuel tests, mechanics, steam engineering, engineering structures, geology, and mining.

Fourth Year.—Mining, metallurgy, economic geology, quantitative analysis of ores, blowpipe analysis, mechanics, designs, estimates and specifications of mining and metallurgical structures, working tests in gold and silver-milling and concentration, electricity, strength of materials, political economy, electricity and magnetism, assaying, engineering, and lithology.

Fifth Year.—Metallurgy, electricity, economic geology, field geology, quantitative analysis, micro-lithology, mining, ore-dressing, mill-work, thermodynamics, mining law, forms and accounts, commercial practice, qualitative analysis, palæontology, and thesis on the establishment and working of mines and smelting-works under given conditions (with drawings, estimates, and written memoirs).

During the summer vacation a school for practical work is held for about two months in some mining district. All the students of the mining department and those about to enter the course of mining and metallurgy after having completed

the two years of preparatory work are required to take part in the work of the summer school. In this way each student receives the benefit of three seasons of practical work in connexion with the course. While in the field the students are under the constant supervision of an assistant, and are required to make complete surface and underground surveys, take sketches and notes of all machinery and appliances used, and as far as possible take part in the practical operations connected with the mining and treatment of ores, etc. During the following term full reports are prepared and handed in, illustrated with accompanying drawings and collections of specimens.

Degrees.

The degrees corresponding to the two courses of study conferred upon the satisfactory completion of the four years' work are :—(1) The degree of bachelor of arts; (2) the degree of bachelor of philosophy.

The degrees granted upon the completion of the several courses of study fall into two classes : professional and non-professional. The only degree conferred upon students in the professional courses (engineering and chemistry) is the professional degree at the completion of the fifth year's work. No professional or master's degree is given at the close of the fourth year.

The degrees corresponding to the courses of study given on the completion of the work as prescribed are : —(*a*) Civil engineer ; (*b*) dynamic engineer ; (*c*) chemist ; (*d*) engineer of mines ; (*e*) bachelor of science ; and (*f*) master of science.

The degree of master of arts is open to all who have received from this university the degree of bachelor of arts.

The degree of master of philosophy is open to all who have received from this university the degree of bachelor of philosophy.

The degree of master of science is open to all who have received from this university the degree of civil engineer, of dynamic engineer, of chemist, or of engineer of mines, as well as to those who shall have pursued the special five years' course prescribed for this degree.

The degree of doctor of philosophy is open to all who have received the degree of master from this university.

In no case will the degree of master, as a second degree, be conferred in less than two years after the date of the lower degree, nor that of doctor in less than one year after the date of the master's degree ; nor will such degrees be recommended except upon satisfactory evidence, to be determined by examination, of a proper amount of study and attainment in advance of undergraduate work.

COLLEGE OF MONTANA, DEER LODGE, MONTANA, UNITED STATES.

The college presents four distinct courses of study, the classical, the scientific, the school of mines, and the English and normal department.

The instruction in the academy is given by the members of the faculties of the college and school of mines. The course of instruction has been prepared with a view to a thorough preparation for the several advanced courses, and at the same time is complete in itself, and well adapted to those who seek only a foundation for the ordinary business of life.

The School of Mines.

The scheme of work for the course in mining engineering is appended ; the course in civil engineering is somewhat similar, but with such modifications as may seem necessary.

The degree of E.M. is given to graduates in the school of mines.

Mining Engineering.

The course of lectures is:—

First Year.—Trigonometry, analytics, general chemistry, German, or some selected language, qualitative analysis, botany, and English.

Second Year.—Analytics, calculus, chemical physics, chemical philosophy, physics, quantitative analysis, descriptive geometry, metallurgy, crystallography, and theoretical mineralogy.

Third Year.—Calculus, mechanics, metallurgy, geology, mechanical engineering, mining engineering, civil engineering, strains in structures, lithology, and mineralogy.

Fourth Year.—Assaying, railroad engineering, strains in structures, ore-dressing, hydraulic engineering, properties of materials, economic geology, applied chemistry, sanitary engineering, heating and ventilation, graphical statics, and lithology.

The practical work during the same period is:—

First Year.—Qualitative chemical analysis and mechanical drawing.

Second Year.—Quantitative chemical analysis, crystallography, and blowpipe analysis.

Third Year.—Mechanical drawing, surveying, blowpipe analysis, and determinative mineralogy.

Fourth Year.—Petrography, railroad surveying, assaying, graphical statics, and engineering designing.

During the vacation at the end of the third year the students in mining engineering are expected to visit a number of mines, and to make a thorough study of at least one or two of them.

During the fourth year each of the students is required to present to the professor of engineering a memoir giving a detailed description of the mines studied during the vacation, and illustrated with drawings carefully made to scale. These memoirs become the property of the engineering department, and are kept for reference purposes.

The advantages of this institution, its various courses of study, are offered to students of both sexes on the same terms. The ladies and gentlemen meet in the dining hall, in the chapel, in the class-rooms, and on such social occasions as the faculty may arrange or approve.

Employment is given to a limited number of students in the care of the buildings and in services in the dining-room, by which part or all of the necessary expenses are met. Rooms may be obtained in town for self-boarding at reasonable rates.

Board at the college includes furnished room, with heat and lights, three meals per day, laundry (not exceeding fifteen pieces per week), mail delivery three times a day, the use of bath-room, reading-room, and library, per month of four weeks, £6. The entire expenses for board and tuition for the college year are £70.

COLUMBIA COLLEGE, CITY OF NEW YORK, UNITED STATES.

Columbia College consists of the school of arts (the original college, founded in 1754); of sundry professional schools, to wit: the school of law, the school of mines, and its medical department by joint resolution, the college of physicians and surgeons, admission to all of which, as candidates for professional degrees, is open to all students whether or not they are college-bred men; and of the univer-

sity faculties of law, mines (mathematics and pure and applied science), political science, and philosophy, which conduct all courses leading to the university degrees of master of arts and doctor of philosophy.

The point of contact between the college and the university is the senior year in the school of arts, during which year students in the school of arts pursue their studies, with the consent of the faculty of arts, under one or more of the university faculties. The various schools are under the charge of their own faculties, and for the better conduct of the strictly university work, as well as of the whole institution, a university council has been established.

The School of Mines.

In the School of Mines, the system of instruction includes:—1. Regular undergraduate courses in (*a*) mining engineering; (*b*) civil engineering; (*c*) electrical engineering; (*d*) metallurgy; (*e*) geology and palæontology; (*f*) analytical and applied chemistry; and (*g*) architecture. 2. Post-graduate courses in (*a*) electrical engineering; (*b*) sanitary engineering; and (*c*) special.

Under the university faculty of mines there is instruction in elective courses for the degree of (*a*) master of arts; (*b*) doctor of philosophy; and (*c*) special courses.

The annual tuition fee for undergraduate students is £40. It is the desire of the trustees to extend, as widely as possible, the educational advantages of the college to deserving young men. Free tuition is therefore offered to undergraduate students under certain conditions which need not be specified here.

Mining Engineering.

The course of study is as follows:—

First Year.—Trigonometry, heat, sound, surveying, botany, chemistry, qualitative analysis, blowpipe analysis, algebra, analytical geometry, descriptive geometry, magnetism, electricity, light, crystallography, drawing, and surveying.

Second Year.—Analytical geometry, calculus, graphics, excavation, biology, hygiene, applied chemistry, mineralogy, tunnelling, drawing, and surveying.

Third Year.—Analytical mechanics, electricity, engineering, properties of materials, graphical statics, mining, geology, assaying, metallurgy, physical laboratory practice, drawing, practical mining, and railroad surveying.

Fourth Year.—Mining, machinery and mill-work, mechanical engineering, heat and its applications, economic geology, metallurgy, ore-dressing, quantitative analysis, assaying, and project or thesis.

Metallurgical Engineering.

The course of study is as follows:—

First Year.—Trigonometry, algebra, heat, sound, magnetism, electricity, light, surveying, chemistry, qualitative and blowpipe analysis, analytical and descriptive geometry, crystallography, drawing, and surveying.

Second Year.—Analytical geometry, calculus, graphics, excavation, tunnelling, hygiene, applied chemistry, quantitative analysis, mineralogy, drawing, and surveying.

Third Year.—Analytical mechanics, electricity and laboratory practice, engineering, properties of materials, graphical statics, mining, geology, quantitative analysis, metallurgy, assaying, drawing, and practical mining.

Fourth Year.—Machinery and mill-work, heat and its applications, economic geology, metallurgy, ore-dressing, quantitative chemical analysis, mining, mechanical engineering, assaying, and project or thesis.

During the vacation at the end of the third year a class in practical mining, composed of students in the course of mining engineering, and in the course of metallurgy, who have completed their third year, is required to visit mines for practical mining work.

During the latter part of the vacations following the close of the first and second years, students in the courses of mining and civil engineering, metallurgy, and geology are required to join the summer class in surveying under the direction of the assistant professor of civil engineering.

During the vacation following the close of the third year students in the courses of mining engineering and metallurgy are required to visit mines and engage in actual work or study under the superintendence of the assistant professor of mining.

Every student who has passed satisfactory examinations is recommended for the degree of engineer of mines, civil engineer, metallurgical engineer, etc.

Degrees.

Any student who has taken his bachelor degree either in Columbia College or in some other college maintaining an equivalent curriculum (every such case of equivalence to be considered on its own merits) shall be entitled, with the approval of the president, to become a candidate for the degrees of master of arts and doctor of philosophy, or either of them.

Each student who declares himself a candidate for the degrees of master of arts and doctor of philosophy, or either of them, shall, immediately upon registration, designate one principal or major subject and two subordinate or minor subjects, which, when approved by the proper faculty, shall be the studies of his university course.

Candidates for the degrees of master of arts and doctor of philosophy, or either of them, must pursue their studies under the direction of the professors and other officers of instruction in charge of the subjects selected by the candidates.

Each candidate for the degree of master of arts, in addition to passing satisfactory examinations on prescribed portions of the subjects selected by him as major and minor, shall present an essay on some topic previously approved by the professor in charge of his major subject.

Each candidate for the degree of doctor of philosophy, in addition to passing satisfactory examinations on the subjects selected by him as major and minor, shall present a dissertation embodying the result of original investigation and research, on some topic previously approved by the faculty.

Every candidate for the degree of doctor of philosophy, in addition to passing such other examinations as may be required by the faculty, shall be subjected to an oral examination on his major subject, and shall defend his dissertation in the presence of the entire faculty, or of so many of its members as may desire to attend. The ability to read at sight two or all of the following languages:—Latin, French, and German—as each faculty may determine, will also be required.

For the degrees of master of arts and doctor of philosophy, or either of them, the course of study shall, so far as possible, embrace instruction in the following groups of subjects:—(*a*) Mathematics; (*b*) mechanics, physics, and chemistry; (*c*) biology, botany, palæontology, mineralogy, lithology, geology, astronomy, meteorology, physical geography, geodesy, and surveying; (*d*) Engineering (civil, mechanical, electrical or sanitary), mining, metallurgy, and architecture. No candidate for a degree may select more than two of his subjects from any one group.

Candidates for the degree of doctor of philosophy, whose preparatory training is purely scientific, will be required to select, for not less than one year, a minor course, under the direction of either the faculty of philosophy or the faculty of political science.

THE OHIO STATE UNIVERSITY, COLUMBUS, OHIO, UNITED STATES.

The university comprises the collegiate department, the law school, and a preparatory department.

The collegiate department embraces schools of arts and philosophy, science, agriculture, engineering, pharmacy, and veterinary medicine.

The instruction given in the collegiate department of the university embraces a wide range of subjects, in accordance with the following classification:—Agriculture, agricultural chemistry, astronomy, botany, civil engineering, drawing, electrical engineering, chemistry, geology, English and rhetoric, French, German, Romance languages, Spanish, Greek, Italian, Latin, history, horticulture, mathematics, mechanical engineering, metallurgy, mine engineering, military science and tactics, pharmacy, philosophy, physics, physiology, political science, veterinary medicine, zoology and entomology.

A charge of £1 a term, or £3 per year, is levied upon all students, under the head of incidental expense. There are dormitories in the university grounds for the use of students at a rent of 6s. per term, and the expense of board, etc., varies from 6s. to 13s. per week. The uniform with which the members of the battalion are required to provide themselves costs about £4 15s. The expenses of a student, excluding clothing (except uniform), and travelling expenses varies from £27 to £75. There is a large amount of work on the university farm which can be performed to advantage by the students, and for which they are paid at current rates for such labour. Some students defray all their expenses in this way.

The School of Engineering.

The school of engineering consists of those departments represented in the courses leading to the degrees of civil engineer, engineer of mines, and mechanical engineer, and in the short course in mining.

No student is permitted to take less than 15 or more than 18 hours a week of class-room work, except by special permission of the committee of the school in which he is enrolled; and no student will be permitted to take more than the regular work of the class to which he belongs, who has not satisfactorily completed all of his work for the preceding term.

The degree of civil engineer is conferred on those who have completed the course of study in civil engineering; that of engineer of mines on those who have completed a course in mining engineering; that of mechanical engineer on those who have completed the course in mechanical engineering or that in electrical engineering.

Mining Engineering.

The course in mining engineering is arranged for students intending to become mining engineers and surveyors, metallurgical or technical chemists. The curriculum, therefore, while keeping mathematics, drawing, and engineering prominent,

also provides extended work in applied chemistry, chemical analysis, assaying, mineralogy, geology, and surveying, with especial application to mines and underground work, while the treatment of ores, both mechanical in ore-dressing and chemical in metallurgy, forms an important feature :—

First Year.—Mathematics, chemistry, mineralogy, drawing, French or German, English, and military drill.

Second Year.—Mathematics, mechanical laboratory work, metallurgy, French or German, English, military drill, and drawing.

Third Year.—Mechanics, strength of materials, physics, metallurgy, bridge-strains, mine-surveying, assaying, determinative mineralogy, and English.

Fourth Year.—Mining engineering, geology, ore-dressing, metallurgy, electrical engineering, photography, and plans and specifications.

As a requisite for graduation each candidate must present an acceptable thesis embodying the results of a special study. The subject of the study must lie within the field of metallurgy or of mining engineering.

Short Course in Mining.

This course is intended for students lacking time and preparation for the full course, and is principally designed for those who have had some practical experience as miners. The work is more elementary in character than in the long course, and is made to apply especially to coal-mining.

First Year.—Mathematics, physical geography, physics, and military drill.

Second Year.—Mine-surveying, ventilation and haulage, mine operating, mineral chemistry, drawing, mechanical laboratory work, geology, and military drill.

THE CASE SCHOOL OF APPLIED SCIENCE, CLEVELAND, OHIO, UNITED STATES.

The object of this school is to teach mathematics, physics, engineering (mechanical and civil), chemistry, economic geology, mining and metallurgy, natural history, drawing, modern languages, and such other kindred branches of learning as the trustees may deem advisable.

The fee for tuition is £15 per year, and the fees for chemicals and use of instruments and apparatus is £5 per year. Young men who are good mechanics, either carpenters or machinists, and who are dependent upon their own exertions for an education, will be furnished work in the school-shops to pay their tuition in part, if they desire it. The cost of text-books, stationery, etc., will average about £5 per year. Board and rooms can be obtained at from 16s. to 24s. per week. The total expense for tuition, board, room, books, etc., for the school year, may vary from £50 to £70.

Courses of instruction are provided in civil engineering, mechanical engineering, electrical engineering, mining engineering, drawing, physics, chemistry, mineralogy, geology, mathematics, astronomy, architecture, and the English, French, and German languages.

The following regular courses of study have been established; each course requires four years for completion, and for proficiency in any of them the degree of bachelor of science in the course pursued is conferred :—(1) General course; (2) civil engineering; (3) mechanical engineering; (4) mining engineering; (5) electrical engineering; (6) physics; (7) chemistry; and (8) architecture.

The general course is intended for students who do not desire to give as much time to a single topic as is given in the other regular courses. During the last two

years the course is largely optional. During the first year the work is the same for all regular students in the school. At the end of this year the student is expected to select one of the regular courses of study to be pursued during the remaining three years of his course.

The course in mining engineering comprises the studies common to all of the engineering courses, and in addition special instruction in mining surveying, mining machinery, chemistry, mineralogy, geology, and metallurgy as follows:—

First Year.—Algebra, chemistry, chemical laboratory work, descriptive geometry, mechanical drawing, trigonometry, analytical geometry, English literature, rhetoric, and French.

Second Year.—Surveying, calculus, physics, physical laboratory work, mechanism, French, German, chemical analysis, surveying and levelling.

Third Year.—Chemical philosophy, chemical analysis, machine design, mining engineering, mechanical laboratory work, mechanics, materials of engineering, physics, physical laboratory work, mineralogy, German, steam engine, topography, geology, and assaying.

Fourth Year.—Engineering construction, metallurgy, practical metallurgy, and ore treatment.

The course is completed by the preparation of a thesis, for which the greater part of the second semester of the fourth year is assigned.

The degree of bachelor of science will be given to those who complete, in a satisfactory manner, either of the regular courses of study. Previous to the conferring of the degree the candidate must prepare and hand in a satisfactory thesis upon some technical subject, selected by him, with the approval of the professor in charge of the department in which he desires to graduate.

The engineering degrees, viz., civil engineer, mechanical engineer, electrical engineer, and engineer of mines, will be conferred upon graduates in those departments who pursue their technical studies one year more, or have been engaged in professional work in positions of responsibility for three years; in either case a further thesis on an entirely original technical topic, or a detailed account or report of the professional work engaged upon, must be presented for acceptance at least twenty days prior to the date of conferring degrees.

UNIVERSITY OF PENNSYLVANIA, PHILADELPHIA, PENNSYLVANIA, UNITED STATES.

The university comprises the following departments:—The college including arts, science, architecture, natural history, finance and economy, and music, and departments of medicine, law, dentistry, philosophy, veterinary medicine, physical education, and hygiene.

The general course in science covers five years—two years of general literary and scientific study, and three years devoted chiefly to technical training in one of the following courses:—Chemistry, mining and metallurgy, civil engineering, and mechanical and electrical engineering.

The work of the first and second years in science includes English, history, mathematics, drawing, chemistry, physics, hygiene, and one modern language (German or French).

In the years devoted to technical instruction are included courses in English, the modern languages, history, philosophy, and political economy, with the necessary instruction in pure and applied mathematics and the scientific branches allied to the principal subject of the course.

Students of regular standing receive at the end of the fourth year the general degree of bachelor of science, and at the end of the fifth year the degree of master of science. The technical degree appropriate to the course pursued may be conferred upon masters of science of two years' standing who have made satisfactory progress in their professions and have presented an acceptable thesis.

Towne Scientific School.

The courses in this school are divided into two groups. In the one are included the courses which are regarded as the continuation of the general course in science; in the other the courses of technical instruction of four years. The technical divisions of the general course in science cover three years.

Good board can be had near the university at from 20s. to 30s. a week. The expenses, including board, tuition, and text-books, will vary from £55 to £95 per annum.

The courses are :—(1) Pure and applied chemistry; (2) metallurgy and mining; (3) civil engineering; and (4) mechanical engineering.

Third Year.—English literature, rhetoric and declamation, German or French, differential and integral calculus, heat and light, electricity and magnetism, mining engineering, mines and mining machinery, prospecting and developing mineral deposits, crystallography, lithology, qualitative chemical analysis, civil engineering, applied mechanics, map projection, graphics, and surveying.

Fourth Year.—English, rhetoric and declamation, economics and social science, experimental physics, physical laboratory work, theory of metallurgical processes, assaying, construction of parts of mines and of mining machinery from notes and sketches, machinery employed in haulage, hoisting, and pumping, mineralogy, gravimetric and volumetric analysis.

Fifth Year.—Metallurgy, mining engineering, geology, and analytical chemistry.

Degrees.

Technical courses of four years lead to the degrees of bachelor of science in mechanical engineering, in electrical engineering, in civil engineering, in chemistry, and in architecture, respectively.

Bachelors of science in mechanical and electrical engineering, civil engineering, architecture, or chemistry, of three years' standing, who have shown marked progress in their professions and have submitted a satisfactory thesis, may receive the degree of master of science, together with the technical degrees appropriate to the course pursued.

The degree of master of arts or of science is conferred upon bachelors of arts or science respectively, on examination after one year of resident study.

THE LEHIGH UNIVERSITY, SOUTH BETHLEHEM, PENNSYLVANIA. UNITED STATES.

The necessary expenses for the collegiate year (clothing and travelling not included) vary from £70 to £100.

The School of Technology.

This school includes seven distinct courses: civil engineering, mechanical engineering, mining, metallurgy, electrical engineering, chemistry, and architecture.

The courses in mining and metallurgy aim at fitting the student for practical work in either of the branches of mining, metallurgy, metallurgical chemistry or geology.

At the end of the first year an opportunity is given the student to select one of two courses leading to the various degrees. These allow a full course in either mining or metallurgy to be acquired in four years, and afford to the student whose time is limited and who desires to practice one of the above branches the means for rapid work. The graduate in either course can obtain the engineer's degree (E.M.) by one year of post-graduate work. For graduates of this university in the course of civil engineering, a one year course has been arranged, leading to the degree of bachelor of science in metallurgy (B.Sc.).

The programme of subjects and studies for the degree of engineer of mines comprise one modern language (French or German), drawing and construction, chemistry, mineralogy, geology, astronomy, applied mechanics, surveying, metallurgy, and mining.

The course of instruction is as follows :—

First Year.—Mathematics, surveying, English, rhetoric, chemistry, French or German, drawing, physiology and health, and gymnasium.

The Course in Metallurgy.

Second Year.—Mathematics, analytical geometry, differential and integral calculus, mechanics, heat, magnetism, electricity, French or German, drawings of metallurgical plant, surveying, chemistry, chemical laboratory work, stoichiometry, English, essays and declamation, and gymnasium.

Third Year.—Mathematics, calculus, analytical mechanics, strength of material, crystallography, assaying, chemical philosophy, French or German, metallurgy, mineralogy, blowpipe analysis, chemistry, quantitative analysis, steam engine, essays and orations, literature and history, and gymnasium.

Fourth Year.—Metallurgy, blowpipe analysis, lithology, mechanics of machinery, quantitative analysis, chemistry, graphical statics, projects in metallurgy, mining, geology, hydraulics, hydrostatics, designing of furnaces and metallurgical plant, American and English literature, Christian evidences, preparation of thesis, and gymnasium.

Fifth Year.—Mining, geology, drawing of mining plant, surveying, astronomy, geological surveying, and preparation of thesis.

The Course in Mining.

Second Year.—Mathematics, analytical geometry, differential and integral calculus, mechanics, heat, magnetism, electricity, French or German, crystallography, surveying, levelling, chemistry and laboratory practice, mineralogy, English, essays, and gymnasium.

Third Year.—Mathematics, calculus, analytical mechanics, strength of materials, geology, lithology, surveying, levelling, blowpipe analysis, steam engine, geological surveying, French or German, literature and history, essays and orations, and gymnasium.

Fourth Year.—Mining, mechanics of machinery, astronomy, surveying, assaying, drawing of mining plant, projects in geology and mining, hydraulics, hydrostatics, American and English literature, Christian evidences, preparation of thesis, and gymnasium.

Fifth Year.—Metallurgy, blowpipe analysis, quantitative analysis, chemical laboratory work, chemical philosophy, drawing, designing furnaces and other metallurgical plant, graphical statics, stoichiometry, projects in metallurgy, astronomy, and preparation of thesis.

Every student will be required to present a thesis upon some topic connected with his special course as a necessary portion of the exercises for his final examination for a diploma. These theses shall be accompanied by drawings and diagrams, when the subjects need such illustration.

Degrees.

On account of the great number and scope of the studies necessary to the attainment of the degree of engineer of mines (E.M.), which includes that of metallurgist, five years are required. At the end of the fourth year the student will have completed a course similar to that leading to the scientific degree in other institutions, and will receive the degree of bachelor of science (B.Sc.).

The faculty will recommend for the degree of master of science any candidate, otherwise properly qualified, who, after taking at this university the degree of bachelor of science, or any degree in the School of Technology, shall pursue, for at least one year at this university, or two years elsewhere, a course of study prescribed by the faculty in at least two departments, pass a thorough examination in the same, and present a satisfactory thesis.

The faculty will recommend for the degree of doctor of philosophy any candidate otherwise properly qualified, who, after taking at this university the degree of master of arts or master of science, shall pursue, for at least one year at this university, or two years elsewhere, a course of advanced study prescribed by the faculty, in at least two departments, pass a thorough examination in the presence of the faculty in the same, and present a satisfactory thesis giving evidence of original investigation. The candidate must have a good knowledge of Latin, and either French or German.

LAFAYETTE COLLEGE, EASTON, PENNSYLVANIA, UNITED STATES.

The aim of Lafayette College is distinctly religious. Under the general direction of the Synod of Pennsylvania of the Presbyterian Church its instruction is in full sympathy with the doctrines of that body. At the same time religious instruction is carried on with a view to a broad and general direction along the lines of general acceptance among evangelical Christians, the points of agreement, rather than those of disagreement, being dwelt upon. Students are expected to attend one of the Presbyterian churches, or a church of the denomination to which they or their parents belong.

The annual college charges are, for those who pay tuition in full, about £23 for technical s udies. With economy, the total annual expenses—exclusive of tuition, clothing, and travelling expenses—need not exceed £50.

Biblical instruction is given in every class of all departments in each week. In the first year a general view of the contents of the Bible and of each book is given, with special attention to chronology, history, and geography. In the second year the Acts of the Apostles is the subject of instruction. In the third year the Epistle to the Romans is studied: the students in the scientific courses read it in German. In the fourth year a course in Christian ethics is given and instruction in Christian evidences. Instruction in the history of the English Bible, its translations and its translators, its merits and its influence, is given from time to time It is intended that the Bible shall be a central object of study throughout the course.

The lectures on health during the first year include the general principles of physiology and anatomy, and special consideration is also given to the bearing of the facts and principles upon natural theology.

Engineer of Mines.

There are three courses in engineering—those of civil, mining, and electrical engineering. The instruction in all the engineering courses is substantially the same during the first two years of the course, except that the electrical engineers substitute the study of physics and practical work in the physical laboratory for a part of the surveying, field, and office-work of the other courses.

The course leading to the degree of engineer of mines is intended to provide in a thorough manner a good foundation for professional work, by a careful study of the facts and principles involved in the numerous problems which are encountered by the mining engineer in the practice of his profession.

First Year.—Mathematics, chemistry, drawing, physiology, French and German, surveying, plane and spherical trigonometry, and mensuration.

Second Year.—Mathematics (analytical geometry, differential calculus, integral calculus), French and German, English, drawing, surveying, mineralogy, practice with blow-pipe, and botany.

Third Year.—Mechanics, surveying, mining, qualitative analysis, heat, light, electricity, descriptive geometry, drawing, quantitative analysis, assaying, optics, acoustics, resistance of materials, elements of machinery, building construction, geology, and New Testament epistles in German.

Fourth Year.—Steam engine, mine-surveying, map of mine-survey, machine drawing and designing, graphical statics, metallurgy, mining, quarrying, political economy, history, quantitative analysis, and graduation thesis.

Degrees.

The degree of civil engineer is conferred on graduates of the civil engineering course; engineer of mines on those of the mining engineering course; and electrical engineer on those of the electrical engineering course.

The degree of master of science may be conferred three years after graduation on any graduate of the scientific department who has passed his graduate period in collegiate or professional study and practice, or who shall submit to the faculty a satisfactory literary, philosophical, or scientific paper. The same degree may be conferred two years after graduation on any graduate of the scientific department who shall have devoted at least one year exclusively to advanced study in the college under the direction of the faculty and passed examinations in the studies pursued.

The degree of doctor of philosophy may be conferred on any graduate of this college who shall have taken a prescribed course of special reading for three years after graduation, passed examinations in approved courses of study, and presented a thesis showing evidence of original research. The same degree may be conferred two years after graduation on any college graduate who, during two years of continuous residence at the college, shall have devoted himself exclusively to advanced studies under the direction of the faculty, passed examinations in them, and presented a satisfactory thesis on one of the studies pursued.

DIAGRAM SHEWING THE HEIGHT OF THE BAROMETER, THE MAXIMA & MINIMA TEMPERATURES & THE DIRECTION OF THE WIND AT THE OBSERVATORIES OF KEW & GLASGOW TOGETHER WITH THE EXPLOSIONS OF FIRE-DAMP IN ENGLAND & SCOTLAND.

1892

JANUARY.

FEBRUARY.

MARCH.

APRIL.

MAY.

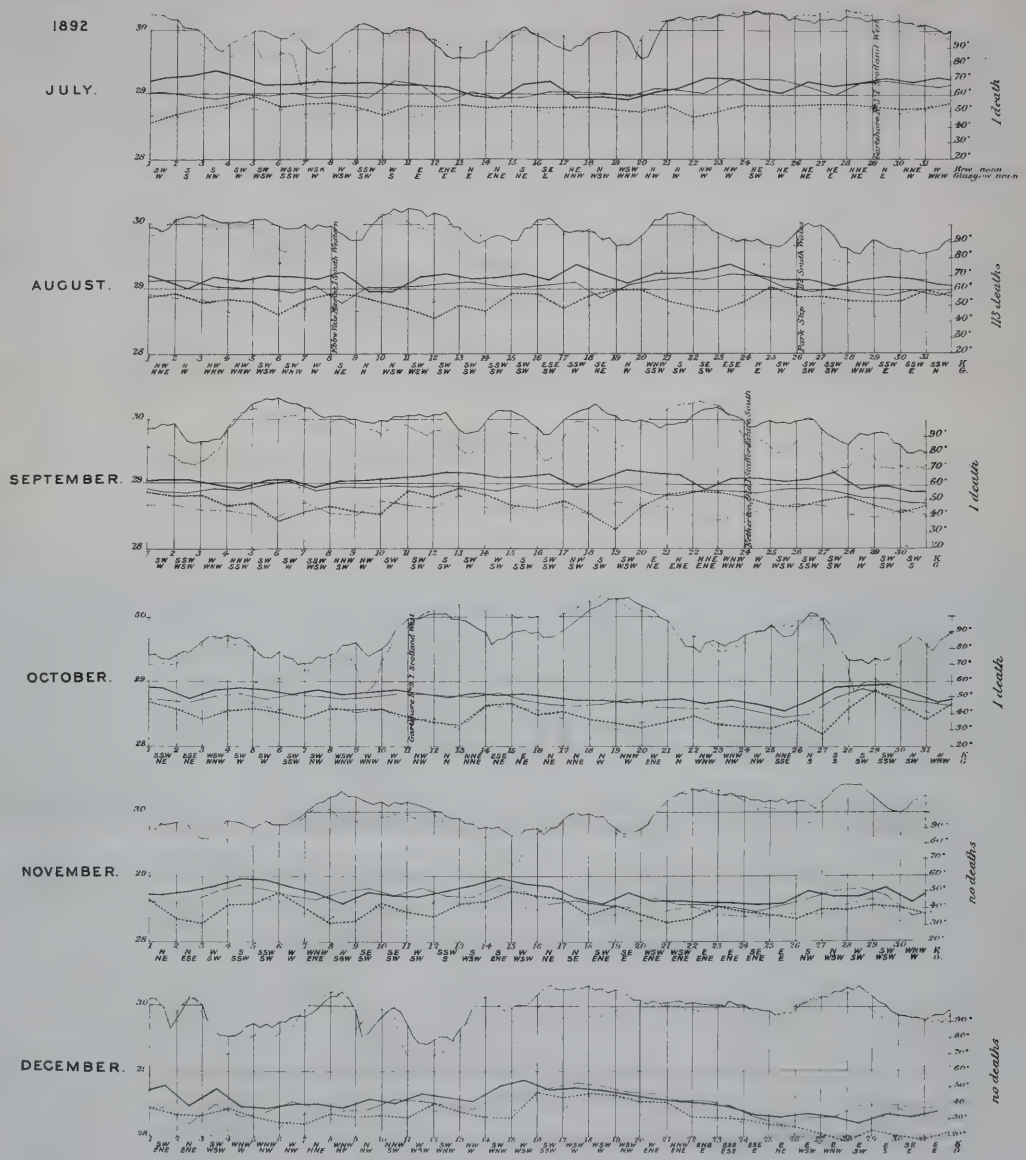
JUNE.

DEATHS FROM EXPLOSIONS OF FIRE-DAMP DURING THE YEAR 1892 IN THE SEVERAL INSPECTION DISTRICTS.

	Deaths
Newcastle	2
Durham	0
Manchester	0
Liverpool	0
Yorkshire	1
Midland	0
Staffordshire North	0
Staffordshire South	2
South Western	1
South Wales	113
Scotland East	2
Scotland West	2
Ireland	0
Total	123

The first top line is the Barometer reading at Kew, taken at 4 a.m. 10 a.m. 4 p.m. & 10 p.m.
The dotted top line is the Barometer reading at Glasgow taken at 4 a.m. 10 a.m. 4 p.m. & 10 p.m.
The lower lines are the Maxima & Minima Temperatures at Kew & Glasgow, observed respectively at midnight & 2 p.m.
Kew lines are black and Glasgow lines are thin.
The figures attached to the districts shew the Number of deaths caused by the Explosions

DIAGRAM SHEWING THE HEIGHT OF THE BAROMETER, THE MAXIMA & MINIMA TEMPERATURES & THE DIRECTION OF THE WIND AT THE OBSERVATORIES OF KEW & GLASGOW TOGETHER WITH THE EXPLOSIONS OF FIRE-DAMP IN ENGLAND & SCOTLAND.



The firm top line is the Barometer reading at Kew, taken at 4 a.m. 10 a.m. 4 p.m. & 10 p.m.
The dotted top line is the Barometer reading at Glasgow taken at 4 a.m. 10 a.m. 4 p.m. & 10 p.m.
The lower lines are the Maxima & Minima Temperatures at Kew & Glasgow, observed respectively at midnight & 4 p.m.
Kew lines are thick, and Glasgow lines are thin.
The figures attached to the districts, shew the Number of Deaths caused by the Explosions.

NORTH OF ENGLAND INSTITUTE

OF

Mining and Mechanical Engineers.

ANNUAL REPORT OF THE COUNCIL

AND

ACCOUNTS FOR THE YEAR 1892-93;

AND LIST OF

COUNCIL, OFFICERS, AND MEMBERS

FOR THE YEAR 1893-94.

1892-93.

NEWCASTLE-UPON-TYNE: PUBLISHED BY THE INSTITUTE.
PRINTED BY ANDREW REID, SONS & CO., LONDON AND NEWCASTLE-UPON-TYNE.

1893.

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Associates	33
Students	34
Subscribers	36

NORTH OF ENGLAND INSTITUTE OF MINING AND MECHANICAL ENGINEERS.

ANNUAL REPORT OF THE COUNCIL.

The following table exhibits the progress of the number of the membership during the two past years :—

	1891.	August 1st. 1892.	1893.
Honorary Members	24	24	25
Members	535	545	565
Associate Members	29	35	47
Associates	28	34	47
Students	36	35	34
Subscribing Collieries	18	18	22
Totals	670	691	740

Ninety-eight members of all classes have joined the Institute during the year, and after allowing for the losses of members through death, etc., there remains a net increase of 49 members. While congratulating the members upon the increasing prosperity of the Institute, the Council would especially ask them at all times to promote its claims and advantages, and to extend its membership.

The connexion with the Federated Institution of Mining Engineers has now existed for four years, and the character of the papers communicated at its well-attended meetings proves that its success is now assured. The Mining Institute of Scotland has now become federated, and it is hoped that the remaining non-federated mining and metallurgical institutes may join at early dates.

The library has been maintained in an efficient condition during the year. The additions by donations, exchange, and purchase have been :—

Bound volumes	260
Pamphlets, reports, etc.	234
A total of	494 titles.

And there are now at least 6,401 volumes and 1,022 unbound pamphlets in the library.

Mr. Thomas Dacres, has presented two MS. volumes of sections of Borings and Sinkings, and the Council have pleasure in recording the thanks of the Institute to him for these valuable records.

The sets of some of the Transactions of Societies, etc., in the library are incomplete owing to loss of volumes or parts, and it is urgently desired that members will return any volumes, etc., in their possession to the librarian.*

Members could also render useful service to the profession, by presentations of books, reports, plans, etc., to the Institute, where they would be preserved in the library, and be available for reference.

The report of the committee on mechanical ventilators is being completed, and the Council trust that it will be shortly issued to the members.

During the course of the year, it was deemed desirable to dissolve the then existing explosives committee, and a new committee was appointed to continue the investigations and to report upon the so-called "flameless explosives." The committee anti ipate being able to issue a preliminary report of their proceedings during the course of the year.

The fifth volume of the Borings and Sinkings is now approaching completion, and the Council trust that the concluding volumes will be issued shortly.

The Institute has again been placed on the list of corresponding societies of the British Association, and Prof. J. H. Merivale has been appointed to act as delegate at the ensuing meeting at Nottingham.

The papers communicated to the Institute have been :—

"Use of Cement in Shaft-sinking." By Mr. Bennett H. Brough.

"Observations on Fans of Different Types working on the same Upcast Shaft." By the Rev. G. M. Capell.

"Manometric Efficiency of Fans." By the Rev. G. M. Capell.

"The Choice of Coarse and Fine-crushing Machinery and Processes of Ore Treatment." By Mr. A. G. Charleton.

"Steam Boilers with Forced Blast: the Perret System for Burning Dust and Rejected Fuels; with notes on Testing Boilers." By Mr. Bryan Donkin, Jun.

"The Electric Transmission of Power and its Analogies in Hydraulics." By Prof. Wm. Garnett.

"Notes on the Occurrence of Manganese Ore near the Arenigs, Merionethshire." By Mr. Edward Halse.

"Notes on the Occurrence of Manganese Ore near Mulegé, Baja California, Mexico." By Mr. Edward Halse.

"The Gold Bearing Veins of the Organos District, Tolima, U.S. Colombia." By Mr. Edward Halse.

* See list in Annual Report of the Council for 1892.

"The Education of Mining Engineers." By Prof. J. H. Merivale.
"The Ordnance Maps." By Prof. J. H. Merivale.
"Joseph Moore's Hydraulic Pumping Arrangement." By Mr. R. T. Moore.
"The Geology and Coal-deposits of Natal." By Mr. R. A. S. Redmayne.
"Gold-mining in Brazil." By Mr. E. M. Touzeau.
"Queensland Coal-mining and the Method adopted to Overcome an Underground Fire." By Mr. E. S. Wight.
"The Coad Electric Miner's Lamp." By Mr. Henry White.

The unfortunate fire in the buildings of the Literary and Philosophical Society resulted in no damage to the buildings of this Institute, which so closely adjoins.

An agreement has been sealed with the North Eastern Railway Company for the use of an entrance from Orchard Street, and the occupation of a broad passage adjoining the basement of the Wood Memorial Hall.

FINANCE REPORT.

The ordinary income for the year 1892-93 amounted to £1,811 14s. 3d., being an increase over the ordinary income of the previous year of £83 18s. 5d.

The total receipts from subscriptions and arrears (excluding sums paid in advance) amounted to £1,460 11s. 0d., an increase of £74 19s. 0d. over the preceding year.

The total amount of subscriptions now in arrear (including those for the year 1892-93) is £220 10s. 0d., which, it is expected, will be fully paid. The sum of £264 12s. 0d., extending over nine years, has been struck off during the year as irrecoverable arrears.

The total expenditure amounted to £1,732 15s. 6d., a decrease of £165 2s. 9d. on the previous year, but that year included exceptional payments on account of cleaning the Wood Memorial Hall, and the expenditure of the Explosives Committee. An increase of £210 11s. 0d. in the subscriptions to the Federated Institution is caused by the increased number of members, and the special call made to meet the deficiency on the first three years' working of that Institution.

DR. THE TREASURER IN ACCOUNT WITH THE NORTH OF ENGLAND FOR THE YEAR ENDING

July 23, 1892.	£	s.	d.	£	s.	d.
To Balance at Bankers	448	14	2			
„ „ in hand	36	18	4			
				485	12	6
„ Dividend of 3¾ per cent. on 134 Shares of £20 each in the Institute and Coal Trade Chambers Co., Ltd., for the half year ending December, 1892	100	10	0			
„ „ „ 3¾ for half year ending June, 1893 ...	100	10	0			
„ Interest on Investments with the River Tyne Commissioners	53	12	6			
				254	12	6
To SUBSCRIPTIONS FOR 1892-93 AS FOLLOWS:—						
431 Members ... ... @ £2 2s.	905	2	0			
26 Associate Members ... ... @ £2 2s.	54	12	0			
28 Associates ... ... @ £1 1s.	29	8	0			
21 Students ... ... @ £1 1s.	22	1	0			
56 New Members ... ... @ £2 2s.	117	12	0			
14 „ Associate Members ... ... @ £2 2s.	29	8	0			
16 „ Associates ... ... @ £1 1s.	16	16	0			
5 „ Students ... ... @ £1 1s.	5	5	0			
	1,180	4	0			
To SUBSCRIBING COLLIERIES, VIZ.:—						
Ashington Coal Company £2 2 0						
Birtley Iron Company 6 6 0						
Bridgewater Trustees 6 6 0						
Marquis of Bute 10 10 0						
Earl of Durham 10 10 0						
Haswell Coal Company 4 4 0						
Hetton Coal Company 10 10 0						
Hutton Henry Coal Company 2 2 0						
Marquess of Londonderry 10 10 0						
North Brancepeth Coal Company ... 2 2 0						
Owners of North Hetton Colliery ... 6 6 0						
Ryhope Coal Company 4 4 0						
Seghill Coal Company 2 2 0						
South Hetton Coal Company 4 4 0						
Stella Coal Company 2 2 0						
Owners of Throckley Colliery 2 2 0						
Victoria Garesfield 2 2 0						
Wearmouth Colliery 4 4 0						
	92	8	0			
„ 1 NEW SUBSCRIBING COLLIERY—						
Butterknowle Coal Company 2 2 0						
	2	2	0			
	1,274	14	0			
Less—Subscriptions for current year paid in advance last year	23	2	0			
	1,251	12	0			
To Arrears	208	19	0			
	1,460	11	0			
„ Subscriptions paid in advance during current year ...	53	11	0			
				1,514	2	0
„ Sale of Publications				42	19	9
				£2,297	6	9

INSTITUTE OF MINING AND MECHANICAL ENGINEERS. CR.
JULY 31, 1893.

July 22, 1893.	£	s.	d.	£	s.	d.
By Stationery Account	157	3	9			
„ Books for Library	74	1	3			
„ Prizes for Papers	36	15	0			
„ Incidental Expenses	53	19	4			
„ Sundry Accounts	14	6	6			
„ Travelling Expenses	0	8	6			
„ Salaries	150	0	0			
„ Clerks' Wages	143	12	8			
„ Reporter's Salary	14	14	0			
„ Rent	95	5	8			
„ Rates and Taxes	17	19	10			
„ Insurance	8	15	9			
„ Furnishing, Repairs, etc.	26	4	9			
„ Coals, Gas, and Water	22	6	0			
„ Postages	45	5	2			
„ General Index	15	19	6			
				876	17	8
„ Fan Committee	2	19	0			
„ Explosives Committee	85	1	1			
„ British Association Meeting—Delegate's Expenses ...	6	9	0			
				94	9	1
By Federated Institution of Mining Engineers—Subscriptions	791	15	10			
Less—Amounts repayable by Authors for Excerpts ...	30	7	1			
				761	8	9
				1,732	15	6
By Balance at Bank	468	3	8			
„ „ in hand	84	12	3			
„ Outstanding Amounts by Authors for Excerpts ...	11	15	4			
				564	11	3
				£2,297	6	9

I have examined the above account with the Books and Vouchers relating thereto, and certify that, in my opinion, it is correct.

JOHN G. BENSON,
CHARTERED ACCOUNTANT.

Newcastle-upon-Tyne,
August 4th, 1893.

DR. THE TREASURER IN ACCOUNT

			£ s. d.	£ s. d.
To 544 Members,				
24 of whom are Life Members.				
520	520	@ £2 2s.		1,092 0 0
To 35 Associate Members.				
5 of whom are Life Members.				
30	30	@ £2 2s.		63 0 0
To 34 Associates		@ £1 1s.		35 14 0
To 37 Students,				
1 transferred to list of Members.				
36	36	@ £1 1s.		37 16 0
To 18 Subscribing Collieries		... ...		92 8 0
To 56 New Members		@ £2 2s.		117 12 0
To 14 New Associate Members		@ £2 2s.		29 8 0
To 16 New Associates		@ £1 1s.		16 16 0
To 5 New Students		@ £1 1s.		5 5 0
To New Subscribing Collieries				6 6 0
				1,496 5 0
To Arrears, as per Balance Sheet 1891-92			470 8 0	
Deduct 6 Students transferred to Members @ £1 1s. ...			6 6 0	
			464 2 0	
Less—Struck off as irrecoverable. Arrears			£206 17 0	
„ „ „ Current year			57 15 0	
			264 12 0	199 10 0
				1,695 15 0
To Subscriptions for previous year paid by New Members				8 8 0
„ Subscriptions Paid in Advance				53 11 0
				£1,757 14 0

WITH SUBSCRIPTIONS, 1892-93. CR.

		PAID. £ s. d.	UNPAID. £ s. d.
By 431 Members, paid	@ £2 2s.	905 2 0	.....
By 87 „ unpaid	@ £2 2s.	.....	182 14 0
By 2 „ dead	@ £2 2s.	.....	4 4 0
520			
By 26 Associate Members, paid	@ £2 2s.	54 12 0	.....
By 4 „ „ unpaid	@ £2 2s.	.....	8 8 0
30			
By 28 Associates, paid	@ £1 1s.	29 8 0	.....
By 6 „ unpaid	@ £1 1s.	.....	6 6 0
34			
By 21 Students, paid	@ £1 1s.	22 1 0	.....
By 15 „ unpaid	@ £1 1s.	.....	15 15 0
36			
By Subscribing Collieries		92 8 0	.....
By 56 New Members, paid	@ £2 2s.	117 12 0	.....
By 14 New Associate Members, paid	@ £2 2s.	29 8 0	.....
By 16 New Associates, paid	@ £1 1s.	16 16 0	.....
By 5 New Students, paid	@ £1 1s.	5 5 0	.....
By 1 New Subscribing Colliery, paid		2 2 0	.....
By 1 „ „ unpaid		.....	4 4 0
		1,274 14 0	221 11 0
Less struck off as irrecoverable		.....	57 15 0
			163 16 0
By Arrears, paid		208 19 0	.....
„ „ unpaid		.....	56 14 0
By Subscriptions paid in advance		53 11 0	.....
		1,537 4 0	220 10 0
			1,537 4 0
			£1,757 14 0

General Statement, July 22nd, 1893.

Liabilities.	£	s.	d.	£	s.	d.
Subscriptions paid in Advance during the year	53	11	0			
„ carried from previous year ...	4	4	0			
				57	15	0
Capital				9,394	8	9
				£9,452	3	9

I have examined the above account with the books and vouchers relating thereto, and certify that, in my opinion, it is correct. The Share Certificates and Bonds have been produced to me.

JOHN G. BENSON,
Chartered Accountant.

Newcastle-upon-Tyne,
August 4th, 1893.

Assets.	£	s.	d.	£	s.	d.
Balance of Account at Bankers	468	3	8			
„ in Treasurer's hands	84	12	3			
Outstanding amounts for Authors' Excerpts ...	11	15	4			
				564	11	3
134 Shares of £20 each in the Institute and Coal Trade Chambers Co., Ltd.	.....			2,680	0	0
Invested with River Tyne Commissioners ...	.....			1,500	0	0
Arrears of Subscriptions	.....			220	10	0
Value of 548 Bound Volumes of Transactions, @ 8s. 6d.	232	18	0			
„ 4,460 Sewn do., @ 6s.	1,338	0	0			
„ 2,421 Unbound Parts, @ 1s.	121	1	0			
„ 77 Copies of Mr. T. F. Brown's Map, @ 2s. 6d.	9	12	6			
„ 375 Copies of Index, Vols. 1-25, @ 1s.	18	15	0			
„ 849 Copies of Catalogue of Fossils, @ 2s. 6d.	106	2	6			
„ 755 Copies of Fossil Illustrations, @ 5s.	188	15	0			
„ 1,500 Copies of Borings and Sinkings (Vol. I. in Sheets) @ 1s.	75	0	0			
„ 275 Do., Vol. I., @ 2s. 6d.... ...	34	7	6			
„ 304 Do., Vol. II., @ 2s. 6d.... ...	38	0	0			
„ 317 Do., Vol. III., @ 2s. 6d.... ...	39	12	6			
„ 429 Do., Vol. IV., @ 2s. 6d.... ...	53	12	6			
„ 263 Copies of Library Catalogue, @ 1s.	13	13	0			
„ 313 Copies of French Commission Report on Explosives in parts, @ 1s. ...	15	13	0			
„ 10 Do., Bound, @ 4s.	2	0	0			
				2,287	2	6
„ Office Furniture and Fittings ...	450	0	0			
„ Books and Maps in Library	1,750	0	0			
				2,200	0	0
				£9,452	3	9

OFFICERS, 1893-94.

PRESIDENT.

A. L. STEAVENSON, Esq., Durham.

VICE-PRESIDENTS.

T. W. BENSON, Esq., 11, Newgate Street, Newcastle-upon-Tyne.
C. BERKLEY, Esq., Marley Hill, Swalwell, R.S.O., Co. Durham.
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G. MAY, Esq., Harton Colliery, near South Shields.

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S. TATE, Esq., Trimdon Grange Colliery, Co. Durham.
R. L. WEEKS, Esq., Willington, Co. Durham.
W. O. WOOD, Esq., South Hetton, Sunderland.

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LINDSAY WOOD, Esq., The Hermitage, Chester-le-Street.

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G. B. FORSTER, Esq., 3, Eldon Square, Newcastle-upon-Tyne.
JOHN DAGLISH, Esq., Rothley Lake, Cambo, R.S.O., Northumberland.
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SECRETARY.

M. WALTON BROWN, Esq., Neville Hall, Newcastle-upon-Tyne.

LIST OF MEMBERS,

JULY 31, 1893.

PATRONS.

His Grace the DUKE OF NORTHUMBERLAND.
The Most Noble the MARQUESS OF LONDONDERRY.
The Right Honourable the EARL OF LONSDALE.
The Right Honourable the EARL OF DURHAM.
The Right Honourable the EARL GREY.
The Right Honourable the EARL OF RAVENSWORTH.
The Right Honourable the EARL OF WHARNCLIFFE.
The Right Reverend the LORD BISHOP OF DURHAM.
The Very Reverend the DEAN AND CHAPTER OF DURHAM.
WENTWORTH B. BEAUMONT, Esq.
BARON BARNARD.

HONORARY MEMBERS.

* Honorary Members during term of office only.

	Date of Election
1 The Right Honourable the EARL of RAVENSWORTH, Ravensworth Castle, Gateshead-upon-Tyne	Nov. 3, 1877
2*J. B. ATKINSON, Esq., H.M. Inspector of Mines, Glasgow ...	Aug. 4, 1888
3*W. N. ATKINSON, Esq., H.M. Inspector of Mines, Newcastle, Staffordshire	Aug. 4, 1888
4*W. BEATTIE-SCOTT, Esq., H.M. Inspector of Mines, Great Barr, near Birmingham	Dec. 10, 1887
5*Prof. P. PHILLIPS BEDSON, Durham College of Science, Newcastle-upon-Tyne	Feb. 10, 1883
6*THOMAS BELL, Esq., H.M. Inspector of Mines, Durham ...	Aug. 4, 1888
7*Prof. G. S. BRADY, Durham College of Science, Newcastle-upon-Tyne	Nov. 6, 1875
8 Dr. BRASSERT, Berghauptmann, Bonn-am-Rhein, Prussia ...	Dec. 8, 1883
9 JOSEPH DICKINSON, Esq., South Bank, Pendleton, Manchester	Dec. 13, 1852
10*C. LE NEVE FOSTER, Esq., Inspector of Mines, Llandudno ..	Aug. 4, 1888
11*Prof. WILLIAM GARNETT, Durham College of Science, Newcastle-upon-Tyne	April 11, 1885
12*JOHN GERRARD, Esq., H.M. Inspector of Mines, Worsley, Manchester	June 11, 1892
13*HENRY HALL, Esq., H.M. Inspector of Mines, Rainhill, Prescott	March 4, 1876
14*J. L. HEDLEY, Esq., H.M. Inspector of Mines, Newcastle-upon-Tyne	April 9, 1892
15*Prof. A. S. HERSCHEL, Observatory House, Slough, Bucks ..	Aug. 3, 1872
16 The Very Rev. Dr. LAKE, Dean of Durham	Feb. 3, 1872
17*Prof. G. A. LEBOUR, Durham College of Science, Newcastle-upon-Tyne	Nov. 1, 1879
18 J. A. LONGRIDGE, Esq., Grêve d'Ayetté, Jersey	Dec. 11, 1886

	Date of Election.
19*J. S. MARTIN, Esq., H.M. Inspector of Mines, Clifton	Aug. 4, 1888
20*JOSEPH T. ROBSON, Esq., H.M. Inspector of Mines, Swansea	Aug. 4, 1888
21*J. M. RONALDSON, Esq., H.M. Inspector of Mines, 44, Athole Gardens, Glasgow	Aug. 4, 1888
22*A. H. STOKES, Esq., H.M. Inspector of Mines, Greenhill, Derby	Aug. 4, 1888
23*PROF. H. STROUD, Durham College of Science, Newcastle-upon-Tyne	Nov. 5, 1892
24 M. E. VUILLEMIN, Mines d'Aniche, Nord, France	Sept. 7, 1878
25*FRANK N. WARDELL, Esq., H.M. Inspector of Mines, Wath-on-Dearne, near Rotherham	Feb. 4, 1865

MEMBERS.

Marked * have paid life composition.

		Date of Election and of Transfer.
1	ABURROW, CHARLES, Box 5, Post Office, Johannesburg, Transvaal	Feb. 13, 1892
2	ACKROYD, WM., Morley Main Collieries, Morley, near Leeds	Feb. 7, 1880
3	AGNIEL, S., Mines de Vicoigne (Nord), Nœux (P. de C.), France	A.M. April 23, 1887 M. June 8, 1889
4	AITKIN, HENRY, Falkirk, N.B....	Mar. 2, 1865
5	ALLAN, JOHN F., c/o Caldwell and Watson, 109, Fenchurch Street, E.C.	A.M. Feb. 10, 1883 M. June 8, 1889
6	ALLAN, T. ALEXANDER, Brightlands, Richmond Hill, Surrey	Feb. 11, 1893
7	ALLISON, J. J. C., Woodland Collieries, Butterknowle, R.S.O., Co. Durham	A.M. Feb. 13, 1886 M. June 8, 1889
8	ANDERSON, C. W., Belvedere, Harrogate	Aug. 21, 1852
9	ANDERSON, R. S., Elswick Colliery, Newcastle-upon-Tyne...	S. June 9, 1883 A.M. Aug. 4, 1888 M. Aug. 3, 1889
10	ANDREWS, HUGH, Swarland Hall, Felton, Northumberland	Oct. 5, 1872
11	ANGUS, JAMES, Radcliffe, Acklington, Northumberland ...	Oct. 8, 1892
12	ARCHER, T., 6, Park Terrace, Gateshead-upon-Tyne ...	July 2, 1872
13	ARMSTRONG, LORD, C.B., LL.D., D.C.L., Cragside, Rothbury (PAST-PRESIDENT, *Member of Council*)	May 3, 1866
14	ARMSTRONG, HENRY, Chester-le-Street *(Member of Council)*	A.M. April 14, 1883 M. June 8, 1889
15	ARMSTRONG, WM., Pelaw House, Chester-le-Street (Retiring VICE-PRESIDENT, *Member of Council)*	Aug. 21, 1852
16	ARMSTRONG, W., Jun., Wingate, Co. Durham	S. April 7, 1867 M. Aug. 6, 1870
17	ARMSTRONG, W. L., Newton Lane Colliery, Victoria Coal and Coke Co., Limited, near Wakefield	S. Mar. 3, 1864 M. Aug. 1, 1868
18	ARNOLD, T., Castle Hill, Greenfields, Llanelly	A.M. Oct. 2, 1880 M. June 8, 1889
19	ASQUITH, T. W., Harperley, Lintz Green, Newcastle-upon-Tyne	Feb. 2, 1867
20	ATKINSON, CLAUDE W., The Electrical Coal Cutting Contract Corporation, Limited, 2, St. Nicholas' Buildings, Newcastle-upon-Tyne	Aug. 6, 1892
21	ATKINSON, FRED. R., Winton Square, Stoke-on-Trent ...	S. Feb. 14, 1874 A.M. Aug. 4, 1883 M. June 8, 1889

Date of Election and of Transfer.

22 Atkinson, L. B., Messrs. T. H. Goolden and Co., Woodfield Works, Harrow Road, London ... ... Aug. 6, 1892

23 Aubrey, R. C., The Midland Coal, Coke and Iron Co., Limited, Halmerend, near Newcastle, Staffordshire ... Feb. 5, 1870

24 Austine, John, Cadzow Coal Co., Glasgow ... . Nov. 4, 1876

25 Ayton, Ernest F., El Bote Mining Negociacion, Zacatecas, Republic of Mexico ... ... Aug. 1, 1891

26 Ayton, Henry, Cowpen Colliery, Blyth, Northumberland S. Mar. 6, 1875
A.M. Aug. 2, 1884
M. June 8, 1889

27 Bailes, E. T., Wingate, Ferryhill ... ... A.M. June 7, 1879
M. June 8, 1889

28 Bailes, T., Jesmond Gardens, Newcastle-upon-Tyne ... Oct. 7, 1858

29 Bailey, Samuel, 30, Waterloo Street, Birmingham ... June 2, 1859

30 Bain, R. Donald, 85, Pembroke Road, Clifton, Bristol ... S. March 1, 1873
M. Aug. 5, 1876

31 Bainbridge, E., Nunnery Colliery Offices, Sheffield ... S. Dec. 3, 1863
M. Aug. 1, 1868

32 Ball, Alfred F., 14, Lansdowne Terrace, Gosforth ... A.M. Dec. 11, 1886
M. June 8, 1889

33 Banks, Thomas, 60, King Street, Manchester ... ... Aug. 4, 1877

34 Barrett, C. R. Whitehill Hall, Chester-le-Street ... ... S. Nov. 7, 1874
A.M. Aug. 7, 1880
M. Dec. 11, 1886

35 Bartholomew, C., Castle Hill House, Ealing, London, W. Aug. 5, 1853

36*Bartholomew, C. W., Blakesley Hall, near Towcester ... Dec. 4, 1875

37 Batey, John, Newbury Collieries, Coleford, Bath ... ... Dec. 5, 1868

38 Baumgartner, W. O., 2, Ash Place, Newcastle Road, Monkwearmouth ... ... S. Sept. 6, 1879
M. Aug. 3, 1889

39 Bayldon, Daniel Hy., 3, Drapers' Gardens, London, E.C.; letters to Spitzkop, Lydenburg, S.A.R. ... ... Feb. 8, 1890

40 Beanlands, Arthur, Palace Green, Durham ... ... Mar. 7, 1867

41 Bell, Sir Lowthian, Bart., Rounton Grange, Northallerton (Past-President, *Member of Council*) ... ... July 6, 1854

42 Bell, C. E., Park House, Durham ... ... S. Dec. 3, 1870
M. Oct. 4, 1879

43 Bell, Geo. F., H. M. Inspector of Mines, 30, Eldon Street, Newcastle-upon-Tyne ... ... S. Sept. 6, 1879
A.M. Aug. 6, 1887
M. Aug. 3, 1889

44 Bennett, Alfred H., Dean Lane Collieries, Bedminster, Bristol ... ... A.M. April 10, 1886
M. June 8, 1889

45 Benson, J. G., 12, Grey Street, Newcastle-upon-Tyne ... Nov. 7, 1874

46 Benson, T. W., 11, Newgate Street, Newcastle-upon-Tyne (Vice-President, *Member of Council*) ... ... Aug. 2, 1866

47 Berkley, C., Marley Hill, Swalwell, R.S.O., Co. Durham (Vice-President, *Member of Council*) ... ... Aug. 21, 1852

48 Berkley, Frederick, Lumley Thicks, Fence Houses ... A.M. Dec. 9, 1882
M. June 8, 1889

49 Berkley, R. W., Marley Hill, Whickham, R.S.O., Co. Durham ... ... S. Feb. 14, 1874
A.M. Aug. 7, 1880
M. June 8, 1889

		Date of Election and of Transfer.
50	BEWICK, T. J., Broad Street House, Old Broad Street, London, E.C.	April 5, 1860
51	BEYNON, J. C. S., P.O. Box 1364, Johannesburg, Transvaal	June 10, 1893
52	BIGGE, D. SELBY, 27, Mosley Street, Newcastle-upon-Tyne...	June 13, 1891
53	BIGLAND, J., Henknowle, Bishop Auckland	June 3, 1857
54	BILGRAMI SYED ALI SHAMSUL ULAMA, Director-General of Mines, Hyderabad, Deccan, India	April 11, 1891
55	BINNS, G. J., Netherseal Colliery, Burton-on-Trent ...	Aug. 7, 1886
56	BIRAM, B., Peaseley Cross Collieries, St. Helen's, Lancashire	Sept. 6, 1855
57	BISHOP, JAMES, Grey Valley Coal Co., Brunnerton, Greymouth, New Zealand	April 13, 1889
58	BLACK, W., Hedworth Villa, South Shields	April 2, 1870
59	BLACKBURN, WM. STEVENSON, Aire Villas, Astley, Woodlesford, near Leeds	Dec. 10, 1887
60	BLACKETT, W. C., Jun., Acorn Close, Sacriston, Durham	S. Nov. 4, 1876
	(Member of Council)	A.M. Aug. 1, 1885
		M. June 8, 1889
61	BLAKELEY, A. B., Soothill Wood Coll. Co., Ltd., near Batley	S. Feb. 15, 1879
		M. Aug. 3, 1889
62	BOLTON, EDGAR ORMEROD, Executor of Col. Hargreaves, Colliery Offices, Burnley	April 12, 1890
63	BOLTON, H. H., Newchurch Collieries, near Manchester ...	Dec. 5, 1868
64	BOUCHER, A. S., P.O. Box 53, Krugersdorf, South African Republic	Aug. 4, 1883
65	BOYD, WM., North House, Longbenton, Newcastle-upon-Tyne	Feb. 2, 1867
66	BRADFORD, GEO., Witton Park, Darlington	Oct. 11, 1890
67	BRAMWELL, HUGH, Great Western Colliery, near Pontypridd, Glamorganshire	S. Oct. 4, 1879
		A.M. Aug. 6, 1887
		M. Aug. 3, 1889
68	BREAKELL, THOMAS, Brassington, near Derby	Feb. 11, 1893
69	BRECKON, J. R., 53, John Street, Sunderland	Sept. 3, 1864
70	BROJA, RICHARD, Kœniglicher Oberbergrath, 35, Friédrich Strasse, Halle, a/S.	Nov. 6, 1880
71	BROUGH, BENNETT H., 5, Robert Street, Adelphi, London, W.C.	A.M. Dec. 10, 1887
		M. June 8, 1889
72	BROUGH, THOMAS, New Seaham Colliery, Sunderland ...	S. Feb. 1, 1873
		A.M. Aug. 2, 1879
		M. June 8, 1889
73	BROWN, M. WALTON, Westmorlands, Gateshead-upon-Tyne	S. Oct. 7, 1871
	(SECRETARY)	M. Aug. 3, 1878
74	BROWN, ROBERT M., Norwood Colliery, *via* Darlington ...	A.M. Oct. 10, 1886
		M. Aug. 3, 1889
75	BROWN, THOMAS, District Manager, Westport Coal Co., Millerton, Westport, New Zealand	April 13, 1889
76	BROWN, THOS. FORSTER, Guildhall Chambers, Cardiff ..	Aug. 1, 1861
77	BROWNE, SIR BENJAMIN C., Westacres, Benwell, Newcastle-upon-Tyne	Oct. 1, 1870
78	BROWNE, R. J. Barakar, East India Railway, Bengal ...	Oct. 8, 1892

			Date of Election and of Transfer.
79	Bruce, John, Port Mulgrave, Hinderwell, R.S.O., Yorkshire	S.	Feb. 14, 1874
		A.M.	Aug. 7, 1880
		M.	June 8, 1889
80	Bryham, William, Rosebridge Colliery, Wigan		Aug. 1, 1861
81	Bryham, W., Jun., Douglas Bank Collieries, Wigan ...		Aug. 3, 1865
82	Bulman, E. H., Shincliffe Rectory, Durham		Feb. 13, 1892
83	Bulman, H. F., Byer Moor, Burnopfield, near Newcastle-upon-Tyne	S.	May 2, 1874
		A.M.	Aug. 6, 1881
		M.	June 8, 1889
84	Bunkell, Henry B., P.O. Box 962, Johannesburg, Transvaal		April 8, 1893
85	Bunning, C. Z., c/o The Borax Co., Limited, 2, Macri Khan, Constantinople	S.	Dec. 6, 1873
		A.M.	Aug. 5, 1882
		M.	Oct. 8, 1887
86*	Burls, Herbert T., Box 76, Barberton, Transvaal, South Africa		Feb. 9, 1889
87	Burn, James, 28, Fawcett Street, Sunderland		Dec. 13, 1890
88*	Burns, David, Canal Bank, Carlisle		May 5, 1877
89	Burrows, J. S., Yew Tree House, Atherton, nr. Manchester	S.	Oct. 11, 1873
		M.	Aug. 4, 1877
90*	Candler, T. E.	S.	May 1, 1875
		A.M.	Aug. 4, 1883
		M.	June 8, 1889
91	Carnes, Charles Spearman, Hutton Henry Colliery, Wingate, R.S.O., Co. Durham		Aug. 1, 1891
92	Chambers, A. M., Thorncliffe Iron Works, near Sheffield...		Mar. 6, 1869
93	Chapman, Alfred Crawhall, 29, St. Nicholas' Buildings, Newcastle-upon-Tyne...		Oct. 13, 1888
94	Charleton, A. G.,		Aug. 6, 1892
95	Charlton, Wm., Linares, Provincia de Jaen, Spain ...		April 8, 1893
96	Cheesman, E. Taylor, Shire Moor Colliery, Northumberland	A.	Aug. 2, 1890
		M.	Aug. 6. 1892
97	Cheesman, Herbert, Hartlepool		Aug. 6. 1892
98	Cheesman, I. T., Throckley Colliery, Newcastle-upon-Tyne		Feb. 1, 1873
99	Cheesman, W. T., Hartlepool		Feb. 5, 1876
100	Chicken, Lancelot W., Boldon Colliery, Co. Durham ...		June 8, 1889
101	Childe, Henry S., Mining Engineer, Wakefield	A.M.	Feb. 12, 1887
		M.	Aug. 3, 1889
102	Clark, C. F., Garswood Coal and Iron Co., Ltd., near Wigan		Aug. 2, 1866
103	Clark, R. B., Springwell Colliery, Gateshead-upon-Tyne ...	S.	May 3, 1873
		M.	Aug. 4, 1877
104	Clough, James, Willow Bridge, Choppington, Morpeth ...	S.	April 5, 1873
		A.M.	Aug. 3, 1878
		M	June 8, 1889
105	Cochrane, B., Aldin Grange, Durham		Dec. 6, 1866
106	Cochrane, C., Green Royde, Pedmore, near Stourbridge ...		June 3, 1857
107	Cochrane, W., St. John's Chambers, Grainger Street West, Newcastle-upon-Tyne (Past-President, *Member of Council*)		Aug. 1, 1861
108	Cockson, Charles, Ince Coal and Cannel Co., Ince, Wigan	A.M.	April 22, 1882
		M.	Aug. 3, 1889
109	Cole, Robert Heath, Endon, Stoke-upon-Trent		Feb. 5, 1876

		Date of Election and of Transfer.
110	COLLINS, ARTHUR LAUNCELOT, 13, Basinghall St., London, E.C.	Feb. 13, 1892
111	COLQUHOUN, T., West Stanley Colliery, Co. Durham ...	Dec. 14, 1889
112	COOK, J., Washington Iron Works, Washington, Co. Durham	May 8, 1869
113	CORBETT, V. W., Chilton Moor, Fence Houses *(Member of Council)*	Sept. 3, 1870
114	CORBITT, M., Teams, Gateshead-upon-Tyne	Dec. 4, 1875
115	CORLETT, GEORGE STEPHEN, Wigan	Dec. 12, 1891
116	COULSON, F., 10, Victoria Terrace, Durham	S. Aug. 1, 1868
		M. Aug. 2, 1873
117	COULSON W., 1, Pimlico, Durham	Oct. 1, 1852
118	COULTHARD, SN. DN. FRANCISCO, Minas del Penoncillo, Marbella, Provincia de Malaga, Spain	June 8, 1889
119	COX, JOHN H., 10, St. George's Square, Sunderland	Feb. 6, 1875
120*	COXE, E. B., Drifton, Jeddo, P.O. Luzerne Co., Penns., U.S.	Feb. 1, 1873
121	CRAVEN, HIRAM, Jun., Mechanical Engineer, Sunderland ...	April 12, 1890
122	CRIGHTON, JOHN, 20, Exchange Buildings, St. Mary's Gate, Manchester	Oct. 8, 1887
123	CRONE, E. W., Forest Hall, near Newcastle-upon-Tyne ...	S. Mar. 5, 1870
		M. Aug. 5, 1876
124	CRONE, F. E.	S. Sept. 2, 1876
		A M. Aug. 7, 1886
		M. June 8, 1889
125	CRONE, J. R., Tudhoe House, *via* Spennymoor	Feb. 1, 1868
126	CRONE, S. C., Forest Hall, Newcastle-upon-Tyne	Aug. 3, 1854
127	CROSS, JOHN, 77, King Street, Manchester	June 5, 1869
128	CROUDACE, C. J., Bettisfield Colliery Co., Limited., Bagillt, North Wales	Nov. 2, 1872
129	CROUDACE, THOMAS, Lambton Lodge, Lambton, Newcastle, New South Wales	Nov. 6, 1862
130	CURRY, W. THOS., Ulverston, Lancashire	S. Sept. 4, 1880
		A.M. Aug. 7, 1886
		M. Aug. 3, 1889
131	DACRES, THOMAS, 20, Princess Street, Bishop Auckland ...	May 4, 1878
132	DAGLISH, JOHN, Rothley Lake, Cambo, R.S.O., Northumberland (PAST-PRESIDENT, *Member of Council*)	Aug. 21, 1852
133	DAKERS, W. R., The Loggins, Tudhoe Colliery, *via* Spenny-	A.M. Oct. 14, 1882
	moor, Durham	M. Aug. 3, 1889
134	DALE, DAVID, West Lodge, Darlington	Feb. 5, 1870
135	DANGAR, J. HUBERT, Camborne, Cornwall	April 8, 1893
136	DARLING, FENWICK, South Durham Colliery, Darlington ...	Nov. 6, 1875
137	DARLINGTON, JAMES, Black Park Colliery, Ruabon, North	S. Nov. 7, 1874
	Wales	M. Aug. 4, 1877
138	DAVEY, GEORGE, Ferro-carril Michoacan y Pacifico, Maravatio, Mexico. Trans. to Las Trojis, Anganguas, Michoacan, Mexico	June 10, 1893
139	DAVEY, HY., 3, Princes Street, Westminster, London, S.W.	Oct. 11, 1873
140	DAVIES, Lt.-Col. JASPER G. S., Marton, Middlesbrough ...	Aug. 3, 1889
141	DAVIES, JOHN, Hartley House, Coundon, Bishop Auckland...	April 10, 1886
142	DAVIES, J. HUBERT, P.O. Box 455, Johannesburg, Transvaal	April 8, 1893
143	DAVIS, KENNETH MCRAE, Dudley Colliery, Northumberland	April 13, 1889
144	DEES, J. GIBSON, Floraville, Whitehaven	Oct. 13, 1883

			Date of Election and of Transfer.
145	Dees, R. R., Newcastle-upon-Tyne		Oct. 7, 1871
146	Denniston, Robert B., Stuart Street, Dunedin, New Zealand	A.M.	Dec. 11, 1876
		M.	Aug. 3, 1889
147	Dixon, D. W., Lumpsey Mines, Brotton, Saltburn-by-the-Sea		Nov. 2, 1872
148*	Dixon, James S., 97, Bath Street, Glasgow		Aug. 3, 1878
149	Dixon, R., 10, Claremont Terrace, Newcastle-upon-Tyne ...		June 5, 1875
150	Dixon-Brown, A. D., 27, Mosley Street, Newcastle-upon-Tyne		Dec. 10, 1892
151	Dodd, B., Bearpark Colliery, near Durham	S.	May 3, 1866
		M.	Aug. 1, 1868
152	Dodd, Cyril H., St. Helens Colliery, Bishop Auckland ...		Aug. 6, 1892
153	Dodd, M., Burncroft, Hexham	S.	Dec. 4, 1875
		A.M.	Aug. 7, 1880
		M.	June 8, 1889
154*	Donkin, W., 17, Havelock Place, Shelton, Stoke-on-Trent	S.	Sept. 2, 1876
		A.M.	Aug. 1, 1885
		M.	June 8, 1889
155	Douglas, C. P., Parliament Street, Consett, Co. Durham ...		Mar. 6, 1869
156	Douglas, John, Seghill Colliery, Dudley, Northumberland	A.M.	April 22, 1882
		M.	Aug. 3, 1889
157	Douglas, M. H., Usworth Colliery, Washington, R.S.O., Co. Durham	A.M.	Aug. 2, 1879
		M.	Aug. 3, 1889
158	Douglas, T., The Garth, Darlington (Vice-President, *Member of Council*)		Aug. 21, 1852
159	Dowdeswell, H., Butterknowle Colliery, *via* Darlington ...	S.	April 5, 1873
		M.	Aug. 7, 1875
160	Doyle, Patrick. *Indian Engineering*. 19. Lall Bazar, Calcutta, India	A.M.	Mar. 1, 1879
		M.	Aug. 3, 1889
161	Eastlake, Arthur W., Balham, London		June 11, 1892
162	Eden, C. H., c/o Messrs. Vivian and Sons, Swansea ...		June 14, 1890
163	Edge, J. H., Coalport Wire Rope and Chain Works, Shifnal, Salop	A.M.	Sept. 7, 1878
		M.	Aug. 3, 1889
164	Elliot, Sir George, Bart., D.C.L., 1, Park Street, Park Lane, London, W. (Past-President, *Member of Council*)		Aug. 21, 1852
165	Ellis, W. R., Wigan		June 1, 1878
166	Elsdon, Robert, The Highlands, Burnt Ash Hill, Lee, Kent		Nov. 7, 1876
167	Elwen, Thomas Lee, Littleburn Colliery, near Durham ...		Oct. 13, 1888
168	Embleton, T. W., The Cedars, Methley, Leeds		Sept. 6, 1855
169	Embleton, T. W., Jun., The Cedars, Methley, Leeds ...	S.	Sept. 2, 1865
		M.	Aug. 1, 1868
170	Eminson, J. B , Londonderry Offices, Seaham Harbour ...		Mar. 2, 1872
171	Everard, J. B., 6, Millstone Lane, Leicester		Mar. 6, 1869
172	Fairley, James, Craghead and Holmside Coll., Chester-le-Street	A.M.	Aug. 7, 1880
		M.	Aug. 3, 1889
173	Faragher, Louis, Cape Copper Co., Limited, Ookiep, Namaqualand, South Africa		April 8, 1893
174	Farmer, A., Seaton Carew, near West Hartlepool		Mar. 2, 1872
175	Faulder, Joseph, Bolton Colliery, Mealsgate, *via* Carlisle...		Aug. 6, 1892
176*	Fenwick, Barnabas, 84, Osborne Road, Newcastle-upon-Tyne		Aug. 2, 1866

		Date of Election and of Transfer.
177	FERENS, FREDERICK J., Silksworth Colliery, Sunderland ...	S. Dec. 4, 1880
		A.M. Aug. 4, 1888
		M. Aug. 3, 1889
178	FERGUSON, D., Persian Bank Mining Rights Corporation Limited., 6, Drapers' Gardens, London, E.C.	A.M. Dec. 8, 1883
		M. Aug. 3, 1889
179	FISHER, ED. R., Nant Glas, Cross Hands, near Llanelly, South Wales	A.M. Aug. 2, 1884
		M. Aug. 3, 1889
180	FISHWICK, R., Binchester Colliery, Co. Durham	Aug. 6, 1892
181	FLEMING, C. E., Black, Hawthorn, & Co., Gateshead-upon-Tyne	April 13, 1889
182	FLETCHER, H., Ladyshore Colliery, Little Lever, Bolton, Lancashire	Aug. 3, 1865
183	FLETCHER, JOHN, Gill House, Ulverstone	S. July 2, 1872
		M. Aug. 4, 1877
184	FLETCHER, LANCELOT, Brigham Hall, Carlisle	A.M. April 14, 1888
		M. June 8, 1889
185	FOGGIN, W., North Biddick Colliery, Washington Station, Co. Durham	Mar. 6, 1875
186	FORREST, J. C., Holly Bank Coll., Essington, Wolverhampton	April 12, 1884
187	FORSTER, G. B., 3, Eldon Square, Newcastle-upon-Tyne (PAST-PRESIDENT, *Member of Council*)	Feb. 5, 1857
188	FORSTER, J. R., Water Company's Office, Newcastle-upon-Tyne	July 2, 1872
189	FORSTER, J. T., Burnhope Colliery, near Lanchester, Co. Durham	S. Aug. 1, 1868
		M. Aug. 5, 1876
190	FORSTER, THOMAS E., 3, Eldon Square, Newcastle-upon-Tyne *(Member of Council)*	S. Oct. 7, 1876
		A.M. Aug. 1, 1885
		M. June 8, 1889
191	FOSTER, GEORGE, Osmondthorpe Colliery, near Leeds ...	Mar. 7, 1874
192	FRYAR, MARK, Denby Colliery, Derby	S. Oct. 7, 1876
		A.M. Aug. 4, 1883
		M. June 8, 1889
193	GALLOWAY, T. LINDSAY, Argyll Colliery, Campbeltown, N.B.	Sept. 2, 1876
194	GALLOWAY, WM., Cardiff	April 23, 1887
195	GALLWEY, ARTHUR P., P.O. Box 138, Johannesburg, Z.A.R.	S. Oct. 2, 1880
		M. Oct. 10, 1891
196	GEDDES, GEORGE H., 142, Princes Street, Edinburgh ...	Oct. 1, 1881
197	GERRARD, JAMES, 11, Meek Buildings, Wigan	S. Mar. 1, 1873
		A.M. Aug. 2, 1879
		M. June 8, 1889
198	GERRARD, JOHN, Worsley, Manchester	S. Mar. 5 1870
		M. Aug. 1, 1874
199	GILCHRIST, J. R., Garesfield Colliery, Lintz Green, Newcastle-upon-Tyne	S. Feb. 3, 1877
		A.M. Aug. 1, 1885
		M. June 8, 1889
200	GILLETT, LEONARD F., 163, Osmaston Road, Derby ...	June 14, 1890
201	GILROY, G., Orrel, Wigan	Aug. 7, 1856
202	GILROY, S. B., 3, Abercrombie Street, Chesterfield	Sept. 5, 1868
203	GJERS, JOHN, Southfield Villas, Middlesbrough	June 7, 1873

			Date of Election and of Transfer.
204	Goolden, Walter T., 28, Westbourne Park, London, W. ...		Aug. 6, 1892
205	Greaves, J. O., St. John's, Wakefield		Aug. 7, 1862
206	Green, J. T., Ty Celyn, Abercarne, Newport, Mon. ...		Dec. 3, 1870
207	Greener, Henry, South Pontop Colliery, Annfield Plain...	A.M.	Dec. 9, 1882
		M.	Aug. 3, 1889
208	Greener, T. Y., West Lodge, Crook, Darlington	S.	July 2, 1872
		A.M.	Aug. 2, 1879
		M.	June 8, 1889
209	Greenwell, G. C., Jun., Poynton, near Stockport	S.	Mar. 6, 1869
		M.	Aug. 3, 1872
210	Gresley, W. S., Reed House, Erie, Pa., U.S.A.	A.M.	Oct. 5, 1878
		M.	Aug. 3, 1889
211	Grey, C. G., 20, Northbrook Road, Leeson Park, Dublin ...		May 4, 1872
212	Griffith, N. R., Plasnewydd, Ruabon, North Wales ...		Aug. 1, 1867
213	Grimshaw, E. J., 23, Hardshaw Street, St. Helen's, Lancashire		Sept. 5, 1868
214	Guthrie, James K., Eltringham Colliery, Prudhoe, R.S.O.		Aug. 3, 1889
215	Haddock, W. T., Jun., The Durban Roodeport Gold Mining Co., Ltd., Roodeport, near Johannesburg, Z.A.R....	S.	Oct. 7, 1876
		A.M.	Aug. 1, 1885
		M.	June 8, 1889
216	Haggie, D. H., Wearmouth Patent Rope Works, Sunderland		Mar. 4, 1876
217	Haggie, Peter Sinclair, Gateshead-upon-Tyne	A.M.	April 14, 1883
		M.	Aug. 3, 1889
218*	Hague, Ernest, Castle Dyke, Sheffield		Aug. 3, 1878
219	Halder, Albert H., Pietersburg, Transvaal		April 8, 1893
220	Hall, M., Lofthouse Station Collieries, near Wakefield ...		Sept. 5, 1868
221	Hall, M. S., 8, Victoria Street, Bishop Auckland		Feb. 14, 1874
222	Hall, Tom, Ryhope Colliery, *via* Sunderland		June 8, 1889
223	Hall, William F., Haswell Colliery, Haswell, *via* Sunderland (Vice-President, *Member of Council*)		May 13, 1858
224	Hallas, G. H., Wigan and Whiston Coal Co., Limited, Prescot	S.	Oct. 7, 1876
		A.M.	Aug. 4, 1883
		M.	June 8, 1889
225	Hallimond, Wm. Tasker, Randt Coal Mining and Land Company, Ld., P.O., Vaal Kranz, *via* Standerton, Transvaal, Z.A.R.		Dec. 14, 1889
226	Halse, Edward, 15, Clarendon Road, Notting Hill, London, W.	A.M.	June 13, 1885
		M.	Aug. 3, 1889
227	Hamilton, E., Rig Wood, Saltburn-by-the-Sea	S.	Nov. 1, 1873
		A.M.	Aug. 2, 1879
		M.	June 8, 1889
228	Hann, Edmund, Aberaman, Aberdare	S.	Sept. 5, 1868
		M.	Aug. 3, 1872
229	Hare, Samuel, Bedlington Collieries, Bedlington, R.S.O....	S.	Aug. 2, 1879
		M.	Aug. 1, 1891
230	Hargreaves, William, Rothwell Haigh, Leeds		Sept. 5, 1868
231	Harle, Peter, Page Bank Colliery, Co. Durham		Oct. 8, 1892
232	Harle, Richard, Browney Colliery, Durham		April 7, 1877
233	Harle, William, Page Bank Colliery, near Durham ...		Oct. 7, 1876
234	Harris, W. S., Kibblesworth, Gateshead-upon-Tyne ...	S.	Feb. 14, 1874
		A.M.	Aug. 7, 1880
		M.	June 8, 1889

		Date of Election and of Transfer.
235	HARRISON, G. B., West Hunwick Colliery, Hunwick, R.S.O., Co. Durham	Aug. 6, 1892
236	HARRISON, W. B., Brownhills Collieries, near Walsall ...	April 6, 1867
237	HAY, J., Jun., Widdrington Colliery, Acklington	S. Sept. 4, 1869
		M. Aug. 4, 1874
238	HAY, WM., Jun , Wood View House, Stanton, Burton-upon-Trent	S. Dec. 8, 1883
		M. Aug. 1, 1891
239	HEAD, JEREMIAH, Queen's Square, Middlesbro'	Aug. 3, 1889
240	HEDLEY, J. L., 22, Hawthorn Terrace, Newcastle-upon-Tyne *(*VICE-PRESIDENT, *Member of Council)*	S. Feb. 5, 1870
		M. Aug. 2, 1873
241	HEDLEY, SEPT. H., Bank Chambers, Wakefield	S. Feb. 15, 1879
		A.M. Aug. 1, 1885
		M. Aug. 3, 1889
242	HEDLEY, W. H., Consett Collieries, Medomsley, Newcastle-upon-Tyne *(Member of Council)*	Aug. 3, 1865
243	HENDERSON, H., Pelton Colliery, Chester-le-Street	Feb. 14, 1874
244	HENDERSON, JOS. J., U.S. Engineer's Office, Kingsbridge, New York, U.S.A.	June 13, 1891
245	HENSHAW, A. MAYON, Talk-o'-th'-Hill Coll., Stoke-upon-Trent	June 8, 1889
246	HEPPELL, T., Leafield House, Birtley, Chester-le-Street *(Member of Council)*...	Aug. 6, 1863
247	HESLOP, C., Upleatham and Lingdale Mines, Upleatham, R.S.O., Yorkshire	S. Feb. 1, 1868
		M. Aug. 2, 1873
248	HESLOP, GRAINGER, Deptford Hall, Sunderland	Oct. 5, 1872
249	HESLOP, THOMAS, Storey Lodge Colliery, Cockfield, *via* Darlington	S. Oct. 2, 1880
		A.M. Aug. 4, 1888
		M. Aug. 3, 1889
250*	HEWITT, G. C., Coal Pit Heath Colliery, near Bristol ...	June 3, 1871
251	HEWLETT, A., Haseley Manor, Warwick	Mar. 7, 1861
252	HIGSON, JACOB, 18, Booth Street, Manchester	Aug. 7, 1862
253	HILL, WILLIAM, Carterthorne Coll. Offices, Witton-le-Wear	A.M. June 9, 1883
		M. Aug. 3, 1889
254*	HILTON, J., 67, Hawkshead Street, Southport	S. Dec. 7, 1867
		M. Aug. 6, 1870
255	HILTON, T. W., Wigan Coal and Iron Co., Limited, Wigan	Aug. 3, 1865
256	HODGSON, JOHN, Edmondsley Colliery, Chester-le-Street ...	Oct. 8, 1889
257	HOLLIDAY, MARTIN F., Langley Grove, Durham ...	May 1, 1875
258	HOLLIS, HENRY WM., Whitworth House, Spennymoor ...	June 8, 1889
259	HOLMES, C., Grange Hill, near Bishop Auckland	April 11, 1874
260	HOMER, CHARLES J., Stoke-on-Trent	Aug. 3, 1865
261	HOOD, A., 6, Bute Crescent, Cardiff	May 2, 1861
262	HOPKINS, EDWARD, 13, Harrington Gardens, London, S.W.	Oct. 8, 1889
263	HOSKOLD, H. D., Inspector General of Mines of the Argentine Republic, and Director of the National Department of Mines and Geology, Casilla Correos, 900, Buenos Ayres	April 1, 1871
264	HOULGATE, J. KERR, 69, Lowther Street, Whitehaven ...	April 8, 1893
265	HURST, GEORGE, 58, Eldon Street, Newcastle-upon-Tyne ...	S. April 14, 1883
		M. Aug. 1, 1891
266	JACKSON, W. G., Hicklam House, Aberford, near Leeds ...	June 7, 1873
267	JARRATT, J., Houghton Main Colliery, near Barnsley ...	Nov. 2, 1867
268	JEFFCOCK, T. W., 18, Bank Street, Sheffield	Sept. 4, 1869

			Date of Election and of Transfer.
269	JENKINS, W., Ocean Collieries, Treorky, Glamorganshire ...		Dec. 6, 1862
270	JENKINS, WM., Consett Iron Works, Consett, Durham ...		May 2, 1874
271	JEPSON, H., 20, The Avenue, Durham	S.	July 2, 1872
		A.M.	Aug. 2, 1879
		M.	June 8, 1889
272*	JOBLING, THOS. E., Croft Villa, Blyth, Northumberland ...	S.	Oct. 7, 1876
		A.M.	Aug. 4, 1883
		M.	June 8, 1889
273	JOHNSON, J., Carlton Main Colliery, Barnsley		Mar. 7, 1874
274	JOHNSON, W., Abram Colliery, Wigan	S.	Feb. 14, 1874
		A.M.	Aug. 2, 1879
		M.	June 8, 1889
275	JOHNSON, WM., Radcliffe Colliery, Acklington, Northumberland		Dec. 9, 1882
276	JOICEY, J. G., Forth Banks West Factory, Newcastle-upon-Tyne		April 10, 1869
277	JOICEY, W. J., Sunningdale Park, Berkshire		Mar. 6, 1869
278	JONES, JACOB CARLOS, Bellambi, New South Wales		Aug. 6, 1892
279	JONES, JOHN ARTHUR, Gijon, Asturias, Spain...		April 8, 1893
280	KAYLL, A. C., Gosforth, Newcastle-upon-Tyne	S.	Oct. 7, 1876
		M.	Aug. 3. 1889
281	KELLETT, WILLIAM, Portland Bank, Southport		June 1, 1878
282	KENDALL, JOHN D., Foxhouses Road, Whitehaven ...		Oct. 3, 1874
283	KIRKBY, J. W., Kirkland, Leven, Fife		Feb. 1, 1873
284	KIRKUP, J. P., Pelton House, Chester-le-Street		April 11, 1891
285	KIRKUP, PHILIP, Cornsay Colliery Office, Esh, near Durham	S	Mar. 2, 1878
		A.M	Aug. 7, 1886
		M.	Aug. 3, 1889
286	KIRTON, HUGH, Kimblesworth Colliery, Chester-le-Street ...	S.	April 7, 1877
		A.M.	Aug. 1, 1885
		M.	June 8, 1889
287	KNOWLES, JOHN, Westwood, Pendlebury, Manchester ...		Dec. 5, 1856
288*	KNOWLES, ROBERT, Ednaston Lodge, near Derby		April 10, 1886
289	LAMB, R., Troughton House, Cleator Moor, *via* Carnforth ..		Sept. 2, 1865
290	LAMB, R. O., West Denton, Newcastle-upon-Tyne		Aug. 2, 1866
291	LAMB, RICHD. W., 29, Great Cumberland Place, London, W.		Nov. 2, 1872
292	LANCASTER, JOHN, Anfield House, Leamington		Mar. 2, 1865
293	LANCASTER, JOHN, Auchinheath, Southfield and Fence Collieries, Lesmahagow		Sept. 7, 1878
294	LANDALE, A., Comely Park Place, Dunfermline		Dec 2, 1858
295*	LAPORTE, H , 57, rue de la Concorde, Brussels		May 5, 1877
296	LAVERICK, JOHN WALES, Tow Law Colliery Office, Tow Law, R.S.O., Co. Durham	A.M.	Dec. 9, 1882
		M.	Aug 3, 1889
297	LAVERICK, ROBT., West Rainton, Fence Houses		Sept 2, 1876
298	LAWRENCE, H., Grange Iron Works, Durham *(Member of Council)*		Aug. 1, 1868
299	LAWRENCE, H. L., Mines de Pierrefitte, Pierrefitte Nestalas, Hautes Pyrenees, France		April 8, 1893
300	LAWS, W. G., Town Hall, Newcastle-upon-Tyne		Oct. 2, 1880

Date of Election and of Transfer.

301 Leach, C. C., Seghill Colliery, Northumberland *(Member of Council)* ... ... S. Mar. 7, 1874
A.M. Aug. 6, 1881
M. Aug. 4, 1883

302 Lebour, G.A., Durham College of Science, Newcastle-upon-Tyne ... ... Feb. 1, 1873

303 Lee, John F., Sheepbridge, Chesterfield .. .. .. A.M. June 13, 1885
M. Aug. 3, 1889

304 Lewis, Sir William Thomas, Mardy, Aberdare .. ... Sept. 3, 1864

305 Liddell, J. M., 3, Victoria Villas, Newcastle-upon-Tyne ... S. Mar. 6, 1875
A.M. Aug. 6, 1881
M. June 8, 1889

306 Linday, James, Bishop Auckland ... ... Aug. 6, 1892

307 Linsley, R., Cramlington Colliery, Northumberland ... July 2, 1872

308 Linsley, S. W., Whitburn Colliery, South Shields ... ... Sept. 4, 1869

309 Lishman, R. R., Durham Main Colliery, Durham ... ... S. June 9, 1883
M. Aug. 1, 1891

310 Lishman, T., Hetton Colliery, Hetton-le-Hole, R.S.O. S. Nov. 5, 1870
(Member of Council) ... ... M. Aug. 3, 1872

311 Lishman, Wm., Holly House, Witton-le-Wear ... ... April 1, 1858

312 Lishman, Wm., Bunker Hill, Fence Houses ... ... Mar. 7, 1861

313 Lisle, J., The Ottos Kopje Diamond Mines, Limited, Ottos Kopje Chambers, Kimberley, South Africa ... ... S. July 2, 1872
A.M. Aug. 3, 1878
M. June 8, 1889

314 Liveing, E. H., 52, Queen Anne Street, Cavendish Square, London, W. ... ... S. Sept. 1, 1877
A.M. Aug. 2, 1884
M. Aug. 3, 1889

315 Livesey, C., Bradford Colliery, near Manchester ... ... Aug. 3, 1865

316 Livesey, T., Bradford Colliery, near Manchester ... ... S. Nov. 7, 1874
M. Aug. 7, 1875

317 Llewellin, David Morgan, Glanwern Offices, Pontypool May 14, 1881

318 Logan, William, Langley Park Colliery, Durham ... ... Oct. 5, 1867

319 Longbotham, Jonathan, Barrow Collieries, Barnsley, Yorkshire ... ... S. May 2, 1868
M. Aug. 2, 1873

320 Longridge, Jethro, Coxlodge Colliery, Newcastle-upon-Tyne ... ... Dec. 14, 1889

321 Louis, D. A., 77, Shirland Gardens, London, W. ... ... April 8, 1893

322 Lowdon, Thos., Hamsteels, near Durham .. .. Dec. 14, 1889

323 Lupton, A., 6, De Grey Road, Leeds ... ... Nov. 6, 1869

324 MacArthur, John S., 12, Knowe Terrace, Pollokshields, Glasgow ... ... April 8, 1893

325 MacCabe, H. O., Russell Vale, Wollongong, New South Wales ... ... S. Sept. 7, 1878
A.M. Aug. 7, 1880
M. Aug. 3, 1889

326 Maddison, Thos. R., Dirtcar House, near Wakefield ... S. Mar. 3, 1877
A.M. Aug. 6, 1881
M. June 8, 1889

327 Maddison, W. H. F., The Lindens, Darlington ... ... June 14, 1890

328 Maling, C. T., Ellison Place, Newcastle-upon-Tyne ... Oct. 5, 1872

329 Mammatt, J. E., St. Andrew's Chambers, Leeds ... ... Aug. 3, 1865

			Date of Election and of Transfer.
330	MARKHAM, G. E., Coundon, Bishop Auckland	S.	Dec. 4, 1875
		A.M.	Aug. 7, 1880
		M.	June 8, 1889
331	MARLEY, J. W., Thornfield, Darlington	S.	Aug. 1, 1868
		M.	Aug. 2, 1873
332	MARTEN, E. B., Pedmore, near Stourbridge		July 2, 1872
333	MARTIN, C. W., Murton Colliery, *via* Sunderland		Aug. 6, 1892
334	MARTIN, T. P., Allhallows Colliery, Mealsgate, Carlisle ...		Feb. 15, 1879
335	MATHIESON, ALEXANDER, Hetton Colliery, Carrington, near Newcastle, New South Wales		Nov. 5, 1892
336	MATTHEWS, J., Messrs. R. and W. Hawthorn, Newcastle-upon-Tyne	A.M.	April 11, 1885
		M.	Aug. 3, 1889
337	MATTHEWS, R. F., Harehope Hall, Alnwick		Mar. 5, 1857
338	MAUGHAN, J. A., Government Central Provinces' Collieries, Umaria, *via* Katni, C.P., India	S.	Nov. 7, 1863
		M.	Aug. 6, 1870
339	MAWSON, R. BRYHAM, Brick House, Westleigh, near Manchester		June 11, 1892
340	MAY, G., Harton Colliery Offices, near South Shields (VICE-PRESIDENT; *Member of Council*)		Mar. 6, 1862
341	MCCARTHY, E. T., Eloiyang, Gleneldon Road, Streatham, S.W., near London	A.M.	Oct. 8, 1887
		M.	Aug. 3, 1889
342	MCCREATH, J., 203, St. Vincent Street, Glasgow		Mar. 5, 1870
343	MCCULLOCH, DAVID, Beech Grove, Kilmarnock, N.B. ...		Dec. 4, 1875
344	MCLAREN, B., Heddon Coal and Fire Brick Co., Wylam-upon-Tyne	S.	Dec. 8, 1883
		A.M.	Aug. 4, 1888
		M.	Aug. 3, 1889
345	MCMURTRIE, GEORGE EDWIN JAMES, Foxes Bridge Colliery, Cinderford, Gloucestershire	S.	Aug. 2, 1884
		M.	Dec. 12, 1891
346	MCMURTRIE, J., Radstock Colliery, Bath		Nov. 7, 1863
347	MERIVALE, Prof. J. H., 2, Victoria Villas, Newcastle-upon-Tyne		May 5, 1877
348*	MERIVALE, W., The Deanery, Ely	A.M.	Mar. 5, 1881
		M.	Aug. 3, 1889
349	MEYSEY-THOMPSON, A. H., Sun Foundry, Leeds	A.M.	April 3, 1889
		M.	Aug. 3, 1889
350	MIDDLETON, ROBERT, Sheep Scar Foundry, Leeds		Aug. 1, 1891
351*	MILLER, N., 31, Hyde Lane, Hyde, near Manchester ...	A.M.	Oct. 5, 1878
		M.	Aug. 3, 1889
352	MITCHELL, CHAS., Jesmond, Newcastle-upon-Tyne		April 11, 1874
353	MITCHELL, CLAYTON E. J., West Highlands, Winchester ...		Nov. 5, 1892
354	MITCHELL, JOSEPH, Mining Offices, Eldon Street, Barnsley		Feb. 14, 1874
355	MITCHINSON, R., Pontop Colliery, Lintz Green Station, Co. Durham		Feb. 4, 1865
356	MITTON, A. DURY, Oakwood, near Cockfield, Co. Durham ...		Aug. 6, 1892
357	MONKHOUSE, JOS., Gilcrux, Cockermouth		June 4, 1863
358	MOOR, WM., Ocean House, Hartlepool	S.	July 2, 1872
		M.	Aug. 4, 1877
359	MOORE, R. T., 156, St. Vincent Street, Glasgow		Oct. 8, 1892
360	MOORE, R. W., Somerset House, Whitehaven	S.	Nov. 5, 1870
		M.	Aug. 4, 1877
361	MOORE, WILLIAM, Loftus Mines, Loftus-in-Cleveland, R.S.O.	A.M.	Nov. 19, 1881
		M.	Aug. 3, 1889

		Date of Election and of Transfer.
362	MOREING, C. A., Broad Street House, Old Broad Street, London, E.C. ... ... A.M.	S. Nov. 7, 1874 Aug. 6, 1881 M. Aug. 3, 1889
363	MORISON, JOHN, Newbattle Collieries, Dalkeith, N.B. ... A.M.	Dec. 4, 1880 M. Aug. 3, 1889
364	MORRIS, WM., Waldridge Colliery, Chester-le-Street ...	Oct. 8, 1892
365*	MORTON, H. J., 2, Westbourne Villas, South Cliff, Scarborough	Dec. 5, 1856
366	MOUNTAIN, WILLIAM C., Forth Banks, Newcastle-upon-Tyne	April 9, 1892
367	MULHOLLAND, M. L., West Cornforth, R.S.O., Co. Durham A.M.	Dec. 11, 1886 M. Aug. 3, 1889
368	MUNDLE, ARTHUR, St. Nicholas' Chambers, Newcastle-upon-Tyne	S. June 5, 1875 M. Aug. 4, 1877
369	MURTON, CHAS. J., Delaval Benwell Colliery, Newcastle-upon-Tyne ... ... A.M.	S. Mar. 6, 1880 Aug. 3, 1887 M. Aug. 3, 1889
370*	NASSE, RUDOLPH, Geheimerbergrat, Dörnsbergstrasse, 6, Berlin, W., Germany	Sept. 4, 1869
371	NEVIN, JOHN, Littlemoor, Mirfield, Normanton	S. May 2, 1868 M. Aug. 5, 1871
372	NEWTON, JAMES, Whitehaven Colliery, Whitehaven ...	Nov. 5, 1892
373	NICHOL, WM., De Beers Mine, Kimberley, South Africa ... A.M.	Oct. 9, 1886 M. Aug. 3, 1889
374	NICHOLSON, J. H., Seaton Delaval Colliery, Newcastle-upon-Tyne	S. Oct. 1, 1881 A. Aug. 3, 1889 M. April 8, 1893
375	NICHOLSON, MARSHALL, Middleton Hall, Leeds	Nov. 7, 1863
376	NORTH, F. W., Rowley Hall Coll., Dudley, Staffordshire ...	Oct. 6, 1864
377	OATES, ROBERT J. W., Rajdoha Mining Company, Limited, *via* Kandra, P.O., Bengal Nagpur Railway, Bengal, India ... ... A.M.	S. Feb. 10, 1883 Aug. 1, 1891 M. Dec. 12, 1891
378	OGILVIE, A. GRAEME, 8, Grove End Road, St. John's Wood, London	Mar. 3, 1877
379	OLDHAM, GEORGE, 25, Western Hill, Durham...	Nov. 5, 1892
380	ORNSBY, R. E., Seaton Delaval Colliery, Newcastle-upon-Tyne ... ... A.M.	S. Mar. 6, 1875 Aug. 4, 1883 M. June 8, 1889
381	PALMER, SIR CHAS MARK, Bart., M.P., Quay, Newcastle-upon-Tyne	Nov. 5, 1852
382	PALMER, A. S., Highfield House, Gateshead	July 2, 1872
383	PALMER, HENRY, East Howle Colliery, near Ferryhill *(Member of Council)* ... ... A.M.	S. Nov. 2, 1878 Aug. 4, 1883 M. Aug. 3, 1889
384	PAMELY, C., 21, Morgan Street, Pontypridd, South Wales	S. Sept. 5, 1868 M. Aug. 5, 1877
385	PANTON, F. S., Silksworth Colliery, Sunderland	S. Oct. 5, 1867 M. Aug, 4, 1874
386	PARRINGTON, M. W., Wearmouth Colliery, Sunderland *(Member of Council)*	S. Dec. 1, 1864 M. Aug. 6, 1870

			Date of Election and of Transfer.
387	Parsons, Hon. Charles Algernon, Elvaston Hall, Ryton-upon-Tyne	A.M.	June 12, 1886
		M.	Aug. 3, 1889
388†	Peace, M. W., Wigan, Lancashire		July 2, 1872
389	Peake, R. C., Cumberland House, Redbourn, Herts.	S.	Feb. 7, 1880
		A.M.	Aug. 7, 1886
		M.	Aug. 3, 1889
390	Pease, Sir J. W., Bart., M.P., Hutton Hall, Guisbrough, Yorkshire		Mar. 5, 1857
391	Peel, Robert, New Brancepeth Colliery, Durham		Aug. 6, 1892
392	Peile, William, Croft Hill, Moresby, Whitehaven	S.	Oct. 1, 1863
		M.	Aug. 6, 1870
393	Pickup, P. W., 71, Preston New Road, Blackburn		Feb. 6, 1875
394	Plummer, John, H.M. Inspector of Mines, Bishop Auckland		June 8, 1889
395	Potter, Addison, C.B., Heaton Hall, Newcastle-upon-Tyne		Mar. 6, 1869
396	Potter, A. M., Riding Mill-upon-Tyne *(Member of Council)*	S.	Feb. 3, 1872
		M.	Aug. 5, 1876
397	Potter, C. J., Heaton Hall, Newcastle-upon-Tyne		Oct. 3, 1874
398	Potts, Jos, Jun., North Cliff, Roker, Sunderland		Dec. 6, 1879
399	Prest, J. J., Shelton Iron, Steel, and Coal Co., Limited, Stoke-on-Trent	S.	May 1, 1875
		A.M.	Aug. 4, 1883
		M.	June 8, 1889
400	Price, J., 6, Osborne Villas, Jesmond, Newcastle-upon-Tyne		Mar. 3, 1877
401	Price, S. R., c/o Messrs. Forster Brown and Rees, Guildhall Chambers, Cardiff	S.	Nov. 3, 1877
		A.M.	Aug. 1, 1885
		M.	Aug. 3, 1889
402	Pringle, Edward, Choppington Colliery, Northumberland		Aug. 4, 1877
403	Pringle, Thomas, Tanfield Lea Colliery, Lintz Green		Aug. 6, 1892
404*	Prior, Edward G., Victoria, British Columbia		Feb. 7, 1880
405	Proctor, C. P., Lightcliffe, near Halifax, Yorkshire	S.	Oct. 7, 1876
		A.M.	Aug. 1, 1885
		M.	June 8, 1889
406	Ramsay, J. A., Sherburn and Littletown Collieries, near Durham		Mar. 6, 1869
407	Ramsay, Wm., Tursdale, Co. Durham		Sept. 11, 1875
		M.	June 8, 1889
408	Redmayne, Richard Augustine Studdert, 26, Grey Street, Newcastle-upon-Tyne	S.	Dec. 13, 1884
		M.	Dec. 12, 1891
409	Reid, Andrew, Newcastle-upon-Tyne		April 2, 1870
410	Reid, Francis, 13, Railway Arches, Westgate Road, Newcastle-upon-Tyne		April 9, 1892
411	Reid, P. S., 20, John Street, Adelphi, London		Dec. 10, 1892
412	Rhodes, C. E., Carr House, Rotherham		Aug. 4, 1883
413	Rich, William, Minas de Rio Tinto, Provincia de Huelva, Spain	A.M.	June 9, 1888
		M.	Aug. 3, 1889
414	Richardson, H., Backworth Colliery, Newcastle-upon-Tyne		Mar. 2, 1865
415	Richardson, Ralph, Whitburn Colliery, South Shields	S.	June 9, 1883
		M.	Aug. 1, 1891
416	Ridley, G., 16, Dean Street, Newcastle-upon-Tyne	S.	Feb. 4, 1865
		M.	Aug. 1, 1868

† Dead.

		Date of Election and of Transfer.
417	RIDYARD, J., Bridgewater Offices, Walkden, near Bolton-le-Moors, Lancashire	Nov. 7, 1874
418	RITSON, U. A., 15, Queen Street, Newcastle-upon Tyne	Oct. 7, 1871
419	ROBERTSON, D. A. W., Metropolitan Colliery, Helensburgh, near Sydney, N.S.W.	Aug. 6, 1892
420*	ROBERTSON, J. R. M., Linton, Mitson's Point, Sydney, N.S.W.	Aug. 2, 1890
421	ROBERTSON, W., 123, St. Vincent Street, Glasgow	Mar. 5, 1870
422	ROBINSON, G. C., Brereton and Hayes Colliery, Rugeley, Staffordshire	Nov. 5, 1870
423	ROBINSON, JOHN THOMAS, Beechburn Colliery, Crook, R.S.O., Co. Durham	Feb. 13, 1892
424	ROBINSON, R., Howlish Hall, near Bishop Auckland *(Member of Council)*	Feb. 1, 1868
425	ROBSON, J. S., Butterknowle Colliery, *via* Darlington	May 15, 1862
426	ROBSON, T. O., Chowdene Cottage, Low Fell *(Member of Council)*	S. Sept. 11, 1875 A.M. Aug. 2, 1884 M. June 8, 1889
427	ROCHESTER, WILLIAM, River View, Blaydon-upon-Tyne	Dec. 13, 1891
428	ROGERSON, JOHN, Croxdale Hall, Durham	Mar. 6, 1869
429	RONALDSON, J. H., Mount Kembla Colliery, Wollongong, N.S.W.	Aug. 6, 1892
430	ROSCAMP, J., Shilbottle Colliery, Lesbury, R.S.O., Northumberland	Feb. 2, 1867
431	ROSS, HUGH, Croxdale Colliery Office, Durham	Aug. 6, 1892
432	ROSS, J. A. G., Newcastle-upon-Tyne	July 2, 1872
433	ROSSER, W., Rhydyrhelig, Sketty, Swansea	Aug. 6, 1857
434	ROTHWELL, R. P., 7, Park Place, New York, U.S.	Mar. 5, 1870
435	ROUTLEDGE, JOS., Greenbank, Chester-le-Street	Sept. 11, 1875
436	ROUTLEDGE, W. H., The Rhyd, Tredegar, Mon., Wales	S. Oct. 7, 1876 A.M. Aug. 1, 1885 M. June 8, 1889
437	ROWAN, JAMES, Inspector of Collieries, Wollongong, N.S.W.	Aug. 6, 1892
438	RUSSELL, ROBERT, Coltness Iron Works, Newmains, N.B.	Aug. 3, 1878
439	RUTHERFORD, W., South Derwent Colliery, Annfield Plain, Lintz Green	Oct. 3, 1874
440	RYDER, W. J. H., Messrs. Mills and Sons, Collingwood Street, Newcastle-upon-Tyne	Nov. 4, 1876
441*	SAISE, WALTER, Manager, E.I.R. Collieries, Giridi, Bengal, India	A.M. Nov. 3, 1877 M. Aug. 3, 1889
442	SAMBORNE, JOHN STUKELY PALMER, Timsbury House, Bath	Aug. 1, 1891
443	SAWYER, A. R., late H.M. Inspector of Mines, c/o Messrs. Thompson, Watson, and Co., Cape Town Transactions to 40, Brompton Square, London, S.W.	S. Dec. 6, 1873 A.M. Aug. 2, 1879 M. June 8, 1889
444	SCARTH, W. T., Raby Castle, Staindrop, Darlington	April 4, 1868
445	SCOTT, ANDREW, Broomhill Colliery, Acklington	Dec. 7, 1867
446	SCOTT, C. F., Grove Cottage, Leadgate, Co. Durham	S. April 11, 1874 M. Aug. 4, 1877
447	SCOTT, ERNEST, Close Works, Newcastle-upon-Tyne	April 9, 1892
448	SCOTT, JOSEPH S., Trimdon Hall, Trimdon Grange, R.S.O.	S. Nov. 19, 1881 M. April 9, 1892

		Date of Election and of Transfer.
449	Scoular, G., Cleator Moor, *via* Carnforth	July 2, 1872
450	Selby, Atherton, Leigh, near Manchester	Oct. 13, 1883
451	Selkirk, John Gunsen, Dalton-in-Furness	April 8, 1893
452	Seymour, L. Irving, De Beers Consolidated Mines, Ld., Kimberley, South Africa	Feb. 14, 1891
453	Shaw, J. Leslie, Somerset House, Whitehaven	Oct. 8, 1892
454	Shaw, W., Wellington Cast Steel Foundry, Middlesbrough	June 3, 1871
455	Shiel, John, Framwellgate Colliery, Co. Durham	May 6, 1871
456	Shipley, T., Castle Eden Colliery, Castle Eden Station ...	A.M. Aug. 2, 1884
		M. Aug. 3, 1889
457	Shone, Isaac, Great George Street Chambers, Parliament Square, London, S.W.	Feb. 3, 1859
458	Shore, Wm. Martin, Kaitangata Railway and Coal Co.'s Collieries, Otago, New Zealand	A.M. April 13, 1889 M. Aug. 3, 1889
459	Shute, C. A., Portland House, Eastbourne, Darlington ...	April 11, 1874
460	Simpson, C. L., Engine Works, Grosvenor Road, Pimlico, London...	April 8, 1893
461	Simpson, F. L. G., Mohpani Coal Mines, Gadawarra, C.P., India	A.M. Dec. 13, 1884 M. Aug. 3, 1889
462	Simpson, F. R., Pelsall Colliery, near Walsall	S. Aug. 4, 1883 M. Aug. 1, 1891
463	Simpson, J., Heworth Colliery, Felling, R.S.O., Co. Durham	S. Dec. 6, 1866 M. Aug. 1, 1868
464	Simpson, J. B., Hedgefield House, Blaydon-upon-Tyne (Past-President, *Member of Council*)	Oct. 4, 1860
465	Simpson, R., Moor House, Ryton-upon-Tyne	Aug. 21, 1852
466	Slinn, T., Plashetts Colliery, Falstone, Northumberland ...	July 2, 1872
467	Smith, Eustace, Newcastle-upon-Tyne	A.M. June 11, 1887 M. Aug. 3, 1889
468*	Smith, R. Clifford, Ashford Hall, Bakewell	Dec. 5, 1874
469	Smith, T. E., Phœnix Foundry, Newgate Street, Newcastle-upon-Tyne	Dec. 5, 1874
470	Sopwith, A., Cannock Chase Collieries, near Walsall ..	Aug. 6, 1863
471	Southern, E. O., Ashington Colliery, near Morpeth ...	S. Dec. 5, 1874 A.M. Aug. 1, 1885 M. June 8, 1889
472	Southern, R., Burleigh House, The Parade, Tredegarville, Cardiff...	Aug. 3, 1865
473	Southworth, Thos., Hindley Green Collieries, near Wigan	May 2, 1874
474	Sparkes, J., Jun., Chester Park, Fishponds, Bristol ...	April 9, 1892
475	Spence, R. F., Backworth, R.S.O., Northumberland ...	S. Nov. 2, 1878 A.M. Aug. 2, 1884 M. Aug. 4, 1889
476	Spencer, Francis H., Robinson Gold Mining Co., Witwatersrand	Dec. 13, 1890
477	Spencer, John, Westgate Road, Newcastle-upon-Tyne ..	Dec. 4, 1869
478	Spencer, John W., Newburn, near Newcastle-upon-Tyne...	May 4, 1878
479	Spencer, T., Ryton, Newcastle-upon-Tyne	Dec. 6, 1866
480	Stansfeld, Harold Sinclair, Flockton Manor, Wakefield	A.M. Oct. 13, 1888 M. Aug. 3, 1889
481	Steavenson, A. L., Durham (President, *Member of Council*)	Dec. 6, 1855

			Date of Election and of Transfer.
482	Stephenson, G. R., 9, Victoria Chambers, Westminster, London, S.W.		Oct. 4, 1860
483	Stevens, A. J., Uskside Ironworks, Newport, Monmouthshire		April 8, 1893
484	Stevens, James, 9, Fenchurch Avenue, London, E.C. ...		Feb. 14, 1885
485	Stobart, F., Biddick Hall, Fence Houses	S.	Aug. 2, 1873
		A.M.	Aug. 5, 1882
		M.	June 8, 1889
486	Stobart, H T., Wearmouth Colliery, Sunderland	S.	Oct. 2, 1880
		A.M.	Aug. 4, 1888
		M.	Aug. 3, 1889
487	Stobart, W., Pepper Arden, Northallerton		July 12, 1872
488	Stobart, Wm. Ryder, Etherley Lodge, Darlington ...		Oct. 11, 1890
489	Stobbs, Jno. T., Walker Colliery, Walker-upon-Tyne ...		Feb. 13, 1892
490	Stoker, Arthur P., Birtley, near Chester-le-Street ...	S.	Oct. 6, 1877
		A.M.	Aug. 1, 1885
		M.	Aug. 3, 1889
491	Storey, Thos. E., Weston Coyney, Longton, Staffordshire		Feb. 5, 1876
492	Straker, J. H., Stagshaw House, Corbridge-upon-Tyne ...		Oct. 3, 1874
493	Stratton, T. H. M., Cramlington House, Northumberland *(Member of Council)*	S.	Dec. 3, 1870
		M.	Aug. 2, 1873
494	Streatfield, Hugh S., Ryhope Colliery, near Sunderland	A.M.	June 8, 1889
		M.	Aug. 3, 1889
495	Swallow, J., Bushblades House, Lintz Green, Newcastle-upon-Tyne		May 2, 1874
496	Swallow, R. T., Wardley Hall, Newcastle-upon-Tyne ...		Feb. 5, 1863
497	Swan, H. F., Beaufront Castle, Hexham		Sept. 2, 1871
498	Swan, J. G., Upsall Hall, near Middlesbrough		Sept. 2, 1871
499	Tate, Simon, Trimdon Grange Colliery, Co. Durham *(Member of Council)*		Sept. 11, 1875
500	Taylor, Hugh, King Street, Quay, Newcastle-upon-Tyne...		Sept. 5, 1856
501	Taylor, T., Quay, Newcastle-upon-Tyne		July 2, 1872
502	Taylor-Smith, Thomas, Broadwood Park, Lanchester ...		Aug. 2, 1866
503	Teasdale, Thomas, Middridge, *via* Heighington, R.S.O. ...		April 9, 1892
504	Telford, W. H., Hedley Hope Collieries, Tow Law, R.S.O., Co. Durham	S.	Oct. 3, 1874
		A.M.	Aug. 6, 1881
		M.	June 8, 1889
505	Thompson, Charles Lacy, Farlam Hall, Milton, Carlisle ...	A.M.	Feb. 10, 1883
		M.	Aug. 3, 1889
506	Thompson, R., Jun., 19, The Crescent, Gateshead-upon-Tyne		Sept. 7, 1867
507	Thompson, W., Ross Buildings, Charters Towers, North Queensland, Australia		Aug. 4, 1888
508	Thomson, John, Eston Mines, by Middlesbrough		April 7, 1877
509	Thomson, Jos. F., Manvers Main Colliery, Rotherham ...		Feb. 6, 1875
510	Tinker, J. J., Hyde, Manchester		April 9, 1892
511	Todd, John T., Bedford Lodge, Bishop Auckland	S.	Nov. 4, 1876
		A.M.	Aug. 1, 1885
		M.	June 8, 1889
512	Touzeau, E. M., Leadenhall Buildings, Leadenhall Street, London. E.C.		Aug. 6, 1892
513	Trelease, W. Henwood, Chief Agent of the Pestarena United G. M. Company, Limited, Pestarena, Vall Anzasca, Novara, Italy		April 8, 1893

			Date of Election and of Transfer.
514*	TYERS, JOHN E., Nerbudda Coal and Iron Company, Limited, Mohpani Coal Mines, *via* Gadawarra, C. Provinces, India	A.M.	Dec. 10, 1877
		M.	Aug. 3, 1889
515	TYZACK, D., 71, Westgate Road, Newcastle-upon-Tyne ...		Feb. 14, 1874
516	VARTY, THOMAS, Skelton Park Mines, Skelton, R.S.O., Cleveland		Feb. 12, 1887
517	VAUGHAN, CEDRIC, Hodbarrow Iron Ore Mines, Millom, Cumberland		Dec. 10, 1892
518	VITANOFF, GEO. N., Sophia, Bulgaria	A.M.	April 22, 1882
		M.	Aug. 3, 1889
519	VIVIAN, JOHN, Vivian's Boring and Exploration Company, Limited, 42, Lowther Street, Whitehaven		Mar. 3, 1877
520	WADDLE, HUGH, Llanmore Iron Works, Llanelly, South Wales		Dec. 13, 1890
521	WADHAM, E., Millwood, Dalton-in-Furness		Dec. 7, 1867
522	WAIN, WM. HOLT, Podmore Hall Colliery, Newcastle-under-Lyne	A.M.	Feb. 12, 1887
		M.	Aug. 3, 1889
523	WALES, H. T., 104, St. Mary Street, Cardiff		Feb. 11, 1893
524	WALKER, J. S., Pagefield Iron Works, Wigan, Lancashire		Dec. 4, 1869
525	WALKER, SYDNEY FERRIS, 196, Severn Road, Canton, Cardiff		Dec. 9, 1882
526	WALKER, W., Hawthorns, Saltburn-by-the-Sea		Mar. 5, 1870
527	WALKER, WM., Jun., H.M. Inspector of Mines, Durham ...		Aug. 3, 1889
528	WALKER, WILLIAM EDWARD, Lowther Street, Whitehaven		Nov. 19, 1881
529	WALLACE, HENRY, Trench Hall, Gateshead-upon-Tyne ...		Nov. 2, 1872
530	WALLAU, JACOB, Messrs. Black, Hawthorn, and Co., Gateshead-upon-Tyne	A.M.	Dec. 10, 1887
		M.	Aug. 3, 1889
531	WALTON, J. COULTHARD, Writhlington Colliery, Radstock, *via* Bath	S.	Nov. 7, 1874
		A.M.	Aug. 6, 1881
		M.	June 8, 1889
532	WARD, H., Rodbaston Hall, near Penkridge, Stafford ...		Mar. 6, 1862
533*	WARD, T. H., Manager Burma Coal Company, Letkobin, Ihingadaw Circle, Shwebo, Upper Burma, East Indies...	A.M.	Aug. 5, 1882
		M.	Aug. 3, 1889
534	WATKYN-THOMAS, W., Mineral Office, Cockermouth Castle	A.M.	Feb. 10, 1883
		M.	Aug. 3, 1889
535	WATSON, THOMAS, Trimdon Colliery, Trimdon Grange ...		Oct. 11, 1890
536	WEBSTER, H. INGHAM, Morton House, Fence Houses ...	A.M.	April 14, 1883
		M.	Aug. 3, 1889
537	WEEKS, J. G., Bedlington, R.S.O., Northumberland ...		Feb. 4, 1865
538	WEEKS, R. L., Willington, Co. Durham *(Member of Council)*	A.M.	June 10, 1882
		M.	Aug. 3, 1889
539	WHITE, C. E., Hebburn Colliery, near Newcastle-upon-Tyne	S.	Nov. 4, 1876
		A.M.	Aug. 1, 1885
		M.	Aug. 3, 1889
540	WHITE, H., Walker Colliery, Newcastle-upon-Tyne ...	S.	Mar. 2, 1867
		M.	Aug. 5, 1871
541	WHITE, J. F., Wakefield	S.	July 2, 1872
		M.	Aug. 2, 1873
542	WHITE, J. W. H., Woodlesford, near Leeds		Sept. 2, 1876
543	WHITEHEAD, JAS., Brindle Lodge, near Preston, Lancashire		Dec. 4, 1875

			Date of Election and of Transfer.
544	WHITELAW, JOHN, 118, George Street, Edinburgh		Feb. 5, 1870
545	WHITTEM, THOS. S., Wyken Colliery, near Coventry ...		Dec. 5, 1874
546	WIDDAS, C., North Bitchburn Colliery, Howden, Darlington		Dec. 5, 1868
547	WIGHT, EDWD. S., Hetton Colliery, Carrington, near New-	A.M.	Dec. 12, 1885
	castle, N.S.W.	M.	Aug. 3, 1889
548	WIGHT, W. H., Cowpen Colliery, Blyth		Feb. 3, 1877
549	WILLIAMS, ERNEST, P.O. Box 965, Bittelheim Buildings, Simmonds Street, Johannesburg, Z.A.R.		Oct. 10, 1891
550	WILSON, J. B., Wingfield Iron Works and Colliery, Alfreton		Nov. 5, 1852
551	WILSON, J. D., Ouston House, Chester-le-Street	S.	Sept. 11, 1875
		A.M.	Aug. 1, 1885
		M.	June 8, 1889
552	WILSON, JOHN ROBINSON, H.M. Inspector of Mines, Leeds	A.M.	June 9, 1883
		M.	Aug. 3, 1889
553	WILSON, ROBERT, Flimby Colliery, Maryport		Aug. 1, 1874
554	WILSON, W. B., Thornley Colliery, by Trimdon Grange, Co. Durham		Feb. 6, 1869
555	WINSTANLEY, ROBT., 28, Deansgate, Manchester		Sept. 7, 1878
556	WINTER, T. B., Grey Street, Newcastle-upon-Tyne... ...		Oct. 7, 1871
557	WOOD, C. L., Freeland, Forgandenny, Perthshire		Aug. 3, 1854
558	WOOD, ERNEST SEYMOUR, South Hetton, Sunderland ...		Oct. 10, 1891
559	WOOD, LINDSAY, The Hermitage, Chester-le-Street (PAST-PRESIDENT, *Member of Council)*		Oct. 1, 1857
560	WOOD, THOMAS, Rainton House, Fence Houses	S.	Sept. 3, 1870
		M.	Aug. 5, 1871
561	WOOD, W. H., Coxhoe Hall, Coxhoe, Co. Durham		Aug. 6, 1857
562	WOOD, W. O., South Hetton, Sunderland *(Member of Council)*		Nov. 7, 1863
563	WORMALD, C. F., Mayfield Villa, Saltwell, Gateshead-upon-	A.M.	Dec. 8, 1883
	Tyne	M.	Aug. 3, 1889
564	WRIGHTSON, T., M.P., Stockton-upon-Tees		Sept. 13, 1873
565	YOUNG, JOHN A., 7, Tyne Vale Terrace, Gateshead-upon-	A.M.	Dec. 10, 1887
	Tyne	M.	Aug. 3, 1889

ASSOCIATE MEMBERS.

Marked * have paid life composition.

		Date of Election.
1	ALLHUSEN, ALFRED, Musgrave House, Gateshead-upon-Tyne ...	Dec. 10, 1892
2	ARMSTRONG, J. H., St. Nicholas' Chambers, Newcastle-upon-Tyne	Aug. 1, 1885
3	ARMSTRONG, T. J., Hawthorn Terrace, Newcastle-upon-Tyne ...	Feb. 10, 1883
4	ATKINSON, G. B., Maritime Buildings, Quay, Newcastle-upon-Tyne	Nov. 5, 1892
5	AUDUS, T., Mineral Traffic Manager, North Eastern Railway, Newcastle-upon-Tyne	Aug. 7, 1880
6	BELL, B. T. A., Secretary of the General Mining Association of the Province of Quebec, Ottawa, Canada	Aug. 6, 1892

Date of Election.

7*Bell, Thos. Hugh, Middlesbrough-upon-Tees ... ... Dec. 9, 1882
8 Bigge, Edward Ellison, General Mining Association, Blomfield House, London Wall, London, E.C. ... ... Aug. 1, 1891
9 Blenkiron, J. Ernest, West House, Arkengarthdale, Richmond, Yorkshire ... ... Feb. 11, 1893
10 Burdon, A. E., Hartford House, Cramlington, Northumberland Feb. 10, 1883

11 Capell, Rev. G. M., Passenham Rectory, Stony Stratford ... Oct. 8, 1892
12*Carr, William Cochran, Benwell Colliery, Newcastle-upon-Tyne ... ... Oct. 11, 1890
13 Cochrane, Henry Heath, Eshwood Hall, Durham ... ... Oct. 10, 1891
14 Cochrane, Ralph D., Hetton Colliery Offices, Fence Houses ... June 1, 1878
15 Cockburn, W. C., 1, St. Nicholas' Buildings, Newcastle-upon-Tyne ... ... Oct. 8, 1887
16 Cooper, R. W., Newcastle-upon-Tyne ... ... Sept. 4, 1880
17 Cornett, J. P., Ford Paper Works, Hylton, Sunderland ... Nov. 5, 1892

18 Dodds, A. P., 13, Dean Street, Newcastle-upon-Tyne ... ... Nov. 5, 1892

19 Edwards, F. H., Forth House, Bewick Street, Newcastle-upon-Tyne ... ... June 11, 1887

20 Fletcher, William, Brigham Hill, *via* Carlisle ... ... Oct. 13, 1883
21 Foster, T. J., Coal Exchange, Scranton, Pennsylvania, U.S. ... Dec. 12, 1891

22 Guthrie, Reginald, Neville Hall, Newcastle-upon-Tyne (Treasurer) ... ... Aug. 4, 1888

23 Hedley, E., Rainham Lodge, The Avenue, Beckenham, Kent ... Dec. 2, 1858
24 Hedley, J. Hunt, John Street, Sunderland ... ... June 13, 1891
25 Hedley, T. F., Jun., Sunderland ... ... April 23, 1887
26 Henderson, C. W. C., The Riding, Hexham ... ... Dec. 9, 1882
27 Henzell, Robt., Close, Newcastle-upon-Tyne ... ... April 11, 1891
28 Hooper, Edward, c/o J. H. Hooper, College Precincts, Worcester June 4, 1881
29 Hopper, J. I., Tyne Dock, South Shields ... ... Dec. 8, 1888
30 Humphreys-Davies, G., 8, Laurence Pountney Hill, Cannon Street, London, E.C. ... ... Oct. 8, 1892

31 Irvine, Joseph, R., Hendon Ropery, Sunderland ... ... Dec. 10, 1887

32*Joicey, James John, Sunningdale Park, Berks. ... ... Oct. 10, 1891

33 Palmer, C. B., Usworth Colliery, Washington, R.S.O. ... ... Nov. 5, 1892
34 Patterson, Wm., Front Street, Tynemouth ... ... Aug. 6, 1892
35*Pease, Arthur, Darlington ... ... Dec. 9, 1882
36 Perkins, Charles. Gallowhill Hall. Newcastle-upon-Tyne ... Aug. 6, 1892
37*Proctor, John H., 29, Side, Newcastle-upon-Tyne .. ... June 8, 1889

38 Richter, F., Osborne Villas, Newcastle-upon-Tyne ... ... Oct. 10, 1891
39 Ridley, J. Cartmell, 3, Summerhill Grove. Newcastle-upon-Tyne Feb. 11, 1893
40 Ridley, Sir Matthew W., Bart., M.P., Blagdon, Northumberland Feb. 10, 1883

Date of Election.

41 Scurfield, George J., Hurworth-upon-Tees, Darlington ... Dec. 9, 1882
42 Shaw, Saville, Durham College of Science, Newcastle-upon-Tyne ... ... April 13, 1889
43 Snowball, Francis John, Seaton Burn House, Dudley, Northumberland ... ... Dec. 12, 1891

44 Todd, James, Elvet Bridge, Durham... ... ... Aug. 6, 1892
45 Topley, William, 28, Jermyn Street, London, S.W. ... ... Oct. 8, 1889

46 Walton, James, 23, Queen Street, Newcastle-upon-Tyne... ... Aug. 6, 1892
47 Wilson, Arthur P., Mansion House Chambers, Queen Victoria Street, London, E.C. ... ... Aug. 3, 1889

ASSOCIATES.

Date of Election and of Transfer.

1 Archer, William, Victoria Garesfield, Lintz Green ... Aug. 6, 1892

2 Barrass, M., Tudhoe Colliery, Spennymoor ... ... S. Feb. 9, 1884
A. Aug. 1, 1891
3 Bates, Sidney, The Grange, Prudhoe-upon-Tyne ... ... Feb. 8, 1890
4 Bell, John, Wardley Colliery, Newcastle-upon-Tyne ... Feb. 8, 1890
5 Berry, Thomas, Bagnall's Houses, Swalwell, R.S.O. ... Nov. 5, 1892
6 Bolam, Philip, North Walbottle Colliery, Newcastle-upon-Tyne ... ... Dec. 12, 1891
7 Bowes, Thomas, Pontop Colliery, Lintz Green Station ... Feb. 13, 1892
8 Buglass, J., Stobswood, *via* Acklington, Northumberland... Dec. 10, 1892

9 Carr, Robert, Usworth Colliery, Washington, R.S.O. ... Nov. 5, 1892
10 Chandley, Charles, Aberdare, South Wales ... ... S. Nov. 6, 1880
A. Aug. 3, 1889
11 Clark, Thomas, Dipton Colliery, Lintz Green Station ... Oct. 11, 1890

12 Draper, William, New Seaham Colliery, Sunderland ... Dec. 14, 1889

13 Fawcett, Edward, Walker Colliery, Walker-upon-Tyne ... June 11, 1892
14 Foster, John William, 24, Silksworth Terrace, New Silksworth, Sunderland ... ... Dec. 14, 1889
15 Fryar, John William, Seghill Colliery, Seghill, Northumberland ... ... June 14, 1890

16 Griffiths, Ed., Brymbo Colliery, near Wrexham, North Wales ... ... June 13, 1891
17 Guy, John George ... ... Aug. 3, 1889

18 Hall, John Charles, Pegswood Colliery, near Morpeth ... Dec. 14, 1889
19 Hepburn, Thomas, Langley Park, Durham ... ... Feb. 8, 1890
20 Hodgson, E., Black Boy Colliery, Bishop Auckland ... Aug. 6, 1892
21 Howes, Frank T., The Poplars, Kingswood Hill, near Bristol Dec. 10, 1892
22 Hunter, Christopher, Broomhill Colliery, Acklington, Northumberland ... ... Dec. 10, 1892

		Date of Election and of Transfer.
23	JOHNSON, WILLIAM, Framwellgate Moor, Durham	Aug. 6, 1892
24	MACKINLAY, EDWARD, East Stanley Colliery, Chester-le-Street	June 11, 1892
25	MACKINTOSH, THOS., Waterloo, Blyth	April 11, 1891
26	MASON, BENJ., Burnopfield, R.S.O., Durham	April 11, 1891
27	MINTO, GEORGE WILLIAM, Heworth Colliery, Heworth-upon-Tyne	Oct. 10, 1891
28	NOBLE, THOMAS GEORGE, Sacriston Colliery, Durham ...	Feb. 13, 1892
29	PARKINSON, W., 6, Ivy Terrace, South Moor, Chester-le-Street	June 13, 1891
30	PEACOCK, THOMAS, Shelton Collieries, Brook Street, Hanley	April 9, 1892
31	PEDELTY, SIMON, Clara Vale, Ryton-upon-Tyne	Dec. 10, 1892
32	RAMSAY, J. G., Page Bank Colliery, Willington, Durham ...	Dec. 10, 1892
33	RICHARDSON, ROBERT, Blaydon Main Colliery, Blaydon-upon-Tyne	Feb. 8, 1890
34	RIDLEY, GEO. D., Tudhoe Colliery, Spennymoor	Feb. 8, 1890
35	RONTREE, THOMAS, Harton Colliery, South Shields ..	Aug. 2, 1890
36	SCOTT, E. CHARLTON, Rainton Colliery, Fence Houses ...	Oct. 8, 1892
37	SCOTT, WM., Thornhill Collieries, near Dewsbury	S. Mar. 4, 1876 A. Aug. 6, 1892
38	SEVERS, WILLIAM, Beamish View, *via* Chester-le-Street ...	Nov. 5, 1892
39	SHAW, JOHN, Cramlington Colliery, Northumberland ...	April 11, 1891
40	SOUTHERN, JOHN, Heworth Colliery, Newcastle-upon-Tyne	Dec. 14, 1889
41	SPRY, JOHN, Murton Colliery, Durham...	Feb. 11, 1893
42	STEAVENSON, C. H., Brotton Mines, Brotton, R.S.O., Yorkshire	S. April 14, 1883 A. Aug. 1, 1891
43	STEPHENSON, WILLIAM, Garesfield Colliery, Lintz Green ...	Dec. 13, 1890
44	THOMPSON, JOSEPH, 38, Church Street, Murton Colliery ...	April 8, 1893
45	WALTON, MATTHEW, Dearham Colliery, Carlisle	Dec. 14, 1889
46	WATSON, JOHN, 19, Bloomfield Terrace, Gateshead-upon-Tyne	Dec. 12, 1891
47	WILSON, R. G., Pelton Colliery, Chester-le-Street	Aug. 6, 1892

STUDENTS.

		Date of Election.
1	ANDREWS, GEO. MURRAY, Broomhill Colliery, Northumberland	Oct. 11, 1890
2	BELL, WALTER, 23, Windsor Terrace, Newcastle-upon-Tyne ...	Oct. 8, 1889
3	BITZOS, N. J., c/o A. G. Sourlas, Balouk-Bazaar, Constantinople	Feb. 14, 1891
4	BROWN, ROBERT O., Elswick Collieries, Newcastle-upon-Tyne ..	Oct. 8, 1892
5	BROWN, WESTGARTH F., Alston House, Cardiff	Aug. 6, 1887
6	BURN, FRANK, H., West Cliff, Elmfield Road, Gosforth, Newcastle-upon-Tyne	Feb. 9, 1889
7	COCHRANE, NAPIER, Aldin Grange, Durham	Dec. 8, 1888

Date of Election.

8 Douglas, David... ... ... Aug. 1, 1891

9 Forster, George W., Newburn-upon-Tyne ... ... Oct. 8, 1887

10 Hall, Fred. W., Haswell Lodge, Sunderland ... ... June 8, 1889

11 Kellett, Matthew H., 5, Elm Street, South Moor, Chester-le-Street ... ... April 11, 1891
12 King, Fred. Cramlington Colliery, R.S.O., Northumberland ... Oct. 8, 1892
13 Kirkup, Austin, Murton Colliery, Sunderland ... ... April 9, 1892
14 Kirkup, Fred. O., Hutton Henry Colliery, Co. Durham ... April 9, 1892

15 Mascall, W. H., c/o Mrs. Race, South Church, near Bishop Auckland ... ... Nov. 5, 1892
16 Mordy, Wm., Shelton Iron, Steel, and Coal Co., Limited, Stoke-on-Trent ... ... April 11, 1891

17 Nicholson, A. D., Elemore Colliery, Fence Houses, Co. Durham June 13, 1885

18 Patterson, Thomas, Craghead, Chester-le-Street ... ... April 13, 1889
19 Potter, E. A., Cramlington, Northumberland ... ... Feb. 6, 1875
20 Pringle, Hy. Géo., Tanfield Lea Colliery, Lintz Green Station, Newcastle-upon-Tyne ... ... Dec. 4, 1880

21 Ridley, Wm., South Tanfield Colliery, Stanley, R.S.O., Newcastle-upon-Tyne ... ... Dec. 9, 1882
22 Ritson, Jno. Ridley, Jesmond Gardens, Newcastle-upon-Tyne... April 11, 1891
23 Robson, William, Walker Colliery, Walker-upon-Tyne ... ... June 11, 1892

24 Simpson, Nelson Ashbridge, Hedgefield House, Blaydon-upon-Tyne ... ... Aug. 1, 1891
25 Stanton, Philip, Throckley Colliery, near Newcastle-upon-Tyne Dec. 8, 1888
26 Sykes, Frank K., Springwell Villa, Bishop Auckland ... ... Feb. 13, 1886

27 Wall, George, South Church, Bishop Auckland ... ... Nov. 5, 1892
28 Watson, Edward, 19, Bloomfield Terrace, Gateshead-upon-Tyne Feb. 13, 1892
29 Weightman, Percy O., Garforth Collieries, near Leeds ... ... Oct. 9, 1889
30 Whitehead, John James, Arden Lea, Heaton, near Bolton ... Dec. 12, 1891
31 Willis, Henry Stevenson, Sacriston, Durham ... ... Feb. 13, 1892
32 Wood, John, Coxhoe Hall, Coxhoe, R.S.O., Co Durham ... June 8, 1889

33 Yeoman, Thomas, Willington Colliery, Co. Durham ... ... Feb. 14, 1885

34 Zumbuloglon, Geo. Chrisostom, No. 3, Astardji han, Grand Bazaar, Constantinople ... ... Feb. 14, 1891

SUBSCRIBERS.

1 Owners of Ashington Colliery, Newcastle-upon-Tyne.
2 Birtley Iron Company (3), Birtley.
3 Bridgewater Trustees (3), c/o Clifford Smith, Esq., Bridgewater Offices, Walkden, Bolton-le-Moors, Lancashire.
4 Marquess of Bute, c/o Messrs. J. and F. Anderson, 48, Castle Street, Edinburgh.
5 Butterknowle Colliery Company, Darlington.
6 Cowpen Coal Company, Limited (2), F, King Street, Newcastle-upon-Tyne.
7 Earl of Durham (5), Lambton Offices, Fence Houses.
8 Elswick Coal Company, Limited, Newcastle-upon-Tyne.
9 Haswell Coal Company (2), Haswell Colliery, Haswell, *via* Sunderland.
10 Hetton Coal Company (5), Fence Houses.
11 Hutton Henry Coal Company, Limited, 7, Bondgate, Darlington.
12 Marquess of Londonderry (5), c/o V. W. Corbett, Esq., Londonderry Offices, Seaham Harbour.
13 North Brancepeth Coal Company, Ltd., Crown Street Chambers, Darlington.
14 Owners of North Hetton Colliery (3), Fence Houses.
15 Ryhope Coal Company (2), Ryhope Colliery, near Sunderland.
16 Owners of Seghill Colliery, Seghill, Northumberland.
17 Owners of South Hetton and Murton Collieries (2), 50, John Street, Sunderland.
18 Owners of Stella Colliery, Hedgefield, Blaydon-upon-Tyne.
19 Owners of Throckley Colliery, Newcastle-upon-Tyne.
20 Owners of Victoria Garesfield Colliery, c/o George Peile, Esq., Shotley Bridge, Co. Durham.
21 Owners of Wearmouth Colliery (2), Sunderland.
22 Westport Coal Company, Limited (2), Manager, Dunedin, New Zealand.

ENUMERATION.

	August 1, 1893.
Honorary Members	25
Members	565
Associate Members	47
Associates	47
Students	34
Subscribers	22
Total	740

Members are desired to communicate all changes of address, or any corrections or omissions in the list of names, to the Secretary.

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